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GEOLOGICAL EVALUATION REPORT

PRE-FEASIBILITY STUDY

HAIL - HARPER CREEK COPPER PROSPECT

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for

AURUN MINES LTD

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Surrey, British Columbia, V3R 1R1

by

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May 1988

ARIS SUMMARY SHEET

District Geologist, Kamloops   Off Confidential: 89.04.21

ASSESSMENT REPORT 17650   MINING DIVISION: Kamloops

PROPERTY: Hail Harper Creek
LOCATION: LAT 51 31 10 LONG 119 49 00
UTM 11 5711134 304581
NTS 082M12W
CLAIM(S): Hail,Judy,Beth,Goof,Sue,Harp,Bob
OPERATOR(S): Aurun Mines
AUTHOR(S): Kaiser, P.B.
REPORT YEAR: 1988, 180 Pages

GEOLOGICAL SUMMARY:

The deposit lies just north of the Cretaceous Baldy Batholith and within metasediments and metavolcanics of the Devonian Eagle Bay Formation. Copper mineralization is confined to tabular-shaped zones within quartz-sericite phyllites and lesser amounts of quartzite. Chaclopyrite occurs as disseminations and patches along foliations, in steeply dipping north striking fractures, within quartz and quartz-carbonate veins and with massive pyrite-pyrrhotite. Sphalerite, galena, arsenopyrite, molybdenite, tentrahedrite-tennanite, bornite, and cubanite are present in minor quantities. Magnetite occurs locally as massive lenses containing minor chalcopyrite.

Reserves in the East zone are estimated at 42,500,000 tonnes grading 0.39 per cent copper, 0.043 grams per tonne gold and 2.4 grams per tonne silver. The West zone contains an estimated 53,500,000 tonnes grading 0.42 per cent copper, 0.047 grams per tonne gold and 2.6 grams per tonne silver.

WORK DONE: Geological
GEOL
MINFILE: 082M 009

1.0 INTRODUCTION

1.1 Terms of Reference

The terms of reference for PBK are contained in Proposal A, dated January 11, 1988. They describe the nature of the work followed to produce this pre-feasibility study.

1.2 Contract

The contract forming the basis for this study was a signed copy of the above proposal and was based on a lump sum price of $50,000.00.

1.3 Purpose

Aurun Mines Ltd. decided to have a pre-feasibility study conducted on its Hail Harper Creek deposit, for which it entered into an option agreement with Quebec Cartier Mines Ltd., a wholly owned subsidiary of U.S. Steel. If positive, Aurun planned to seek a partner to proceed to the next stage, which would be a definitive feasibility study.

1.4 Approach used

By definition, a pre-feasibility study does not attempt to optimize any one part of its scope, but rather, using the judgment of the staff assigned to it, makes the best guess at all the items it covers. By necessity, due to time and financial constraints, these decisions must be made quickly, and it is not possible to dwell on them at length.

AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
PRE-FEASIBILITY STUDY
FINAL REPORT

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2.0 CONCLUSIONS AND RECOMMENDATIONS

2.1 Conclusions

2.1.1 Geological

The ore reserves and grades within the U.S. Steel mineral claim boundaries under option by Aurun Mines Ltd. (East Zone), which were the area under review by this study combined with Noranda's (West Zone) claims, were found to be as follows:

	Geological Ore Reserves (metric tonnes)	Grades		
		Cu %	Au g/t	Ag g/t
East Zone	42,500,000	0.39	0.043	2.4
West Zone	53,500,000	0.42	0.047	2.6
Total	**96,000,000**	**0.41**	**0.045**	**2.5**

It is believed that the chances of expanding these reserves are positive, and that further expenditures on geological exploration are justified.

2.1.2 Mining

Mineable ore reserves and grades within the same areas as described under Geological (Section 2.1.1) are summarized as follows:

	Mineable Ore Reserves (metric tonnes)	Grades Cu %	Au g/t	Ag g/t
East Zone	42,200,000	0.34	0.037	2.1
West Zone	23,140,000	0.40	0.044	2.4
Total	**65,340,000**	**0.36**	**0.040**	**2.2**

At the daily production rate of 12,600 tonnes per day, the above reserves will sustain production for over 10 years of continuous operation on the basis of 350 days per year and 24 hours per day.

The ore can be sequentially mined from two open pits using conventional truck and shovel techniques within these pits.

2.1.3 Processing Plant

Production of a saleable concentrate can be achieved using a standard flowsheet and conventional equipment, similar to plants such as Brenda Mines. This plant will process 12,600 tonnes of ore per day.

2.1.4 Infrastructure

The status of the key aspects of infrastructure for this project is as follows:

Site selection - a suitable site exists one half a kilometre north from the mid-point of the two ore zones.

Power supply - power for the project will come from the B.C. Hydro grid and will require the construction of a power line 11 kilometres long.

Water supply - this is anticipated to come from the North Thompson River along a pipeline eight kilometres long with a head of 1200 metres.

Waste water - this will be disposed of in a settling ponds located one half to two and a half kilometres from the site. Reclaim water will be pumped back to the site.

Tailings pond - this will be located three kilometres west of the plant and will have a capacity of 35 million cubic metres, sufficient to handle the tailings at the recommended production rate from the plant.

Access road - this follows the existing road widening it to 20 metres and improving its construction. It is 14 kilometres long with a 10% grade.

2.1.5 Cost Estimates

2.1.5.1 Capital Cost Estimates

Two cases are presented - the Base Case of $145,643,000 and the Revised Case of $133,886,000.

These are summarized by work area as follows:

Cost Item	Base Case X $1,000	Revised Case X $1,000
Preproduction stripping	1,400.	1,400.
Access Road	1,729.	1,729.
Tailings/Mine Waste	4,068.	4,068.
Site and Utilities	17,895.	17,895.
Mine Equipment	16,780.	16,780.
Milling	42,994.	42,994.
Pre-Op Testing, Spares	1,684.	1,684.
Sub-Total	86,550.	86,550.
Taxes, Escalation	17,347.	14,865.
Construction Overhead	12,356.	7,423.
EPCM	10,390.	7,585.
Contingency	19,000.	17,463.
TOTAL	**$145,643.**	**$133,886.**

In keeping with the parameters set out for this study, the above estimate includes a contingency of 15% of total direct, indirect construction, engineering procurement and construction management costs.

2.1.5.2 Operating Cost Estimates

These are summarized by work function as follows:

	$ per tonne of ore
Mining	3.3395
Processing	2.7475
Plant Services	0.3306
Administration	0.1941
TOTAL	**6.6117**

Operating expenses also include depreciation based on capital at 30% of declining balance and royalty at 3 percent of total sales revenue.

2.1.5.3 Financial Analysis

Based on the current financial model used for this study, which assumes both the Noranda and Quebec Cartier ore bodies are mined, the rate of return on investment is negative. This financial model assumes copper prices of $1.08 (U.S.) per pound and used equipment (75% of new equipment costs). Financial models have been included covering both new and used equipment for copper prices of up to $2.00 (U.S.) per pound.

2.1.6 Future Work Programs

In order for production to proceed in an orderly fashion, the work programs outlined below should commence as soon as possible.

2.1.6.1 Geological

In order to provide drilling data on which to base ore reserve calculations of the accuracy required for a definitive feasibility study, it will be necessary to spend $341,000. These results will then be analysed and interpreted with the objective of increasing the metal units, thus improving the economics of the project.

2.1.6.2 Mine Planning

Using the geological data derived in the above step, an optimum pit design will be drawn of a quality which will permit mine planning to proceed.

2.1.6.3 Infrastructure

All aspects of this work area will be detailed to the level required to allow permitting, meeting all the necessary environmental requirements and applications for surface development, allowing production to start as scheduled.

2.1.6.4 Cost Estimates

In keeping with the accepted definition of contingency required for a definitive feasibility study of a type recognized by financial institutions, these estimates will be ± 10% accurate for capital costs, and ± 5% for operating costs.

2.2 Recommendations

Based on this study, it is recommended that the project proceed provided two main factors are improved as follows:

2.2.1 Metal Units (grade/tonnage)

These (particularly copper) will have to be increased, if the price for them is not higher than the following:

(Current March/88 levels)

Copper	$ 1.08	(U.S.)	per pound
Gold	$456.00	(U.S.)	ounce
Silver	$ 6.69	(U.S.)	ounce

The key to this required increase will be a more detailed geological approach and optimum mine planning effort, both with the intent of maximizing the units of the various metals. Financial evaluations for copper prices of up to $2.00/lb (U.S.) are included in the report.

2.2.2 Capital Costs

Of equal importance to this project will be the ability of the Owner to source good, used equipment for as many aspects of the project as possible.

3.0 GEOLOGICAL DATA

3.1 General Description

The Hail Harper Creek copper prospect is located within the Adams Lake Map Area, British Columbia, at the headwaters of Harper Creek. It is about 7 air miles southeast of Vavenby, in the valley of the North Thompson River, and about 75 air miles north from Kamloops.

Access to the property is by the main line of the Canadian National Railway or British Columbia Highway 5, both routes following the North Thompson Valley. The prospect was discovered and extensively trenched and drilled during the late 1960's and early 1970's by Quebec Cartier Mining Company (East Zone) and by Noranda Exploration Company (West Zone). Aurun Mines Ltd. has entered an option agreement with Quebec Cartier Mining Company for that company's mineral interest in the area.

The Hail Harper Creek copper prospect is situated about 3,000 metres north of the Cretaceous quartz monzonite Baldy Batholith. It lies within metavolcanics and metasediments of the Eagle Bay Formation, probably Paleozoic in age. A hornfels zone has developed with a variable width in the periphery of the batholith, south of the copper prospect. The rocks hosting the copper prospect are located in the north limb of a broad synform oriented along the North Thompson River Valley north of the property.

Outcrop on the property is generally less than 5 percent. The overburden consists of a generally thin mantle of glacial debris and semi-residual rubble.

Thickness of the overburden on the east side of the property is:

average:	4.6 m (σ 2.50)
minimum:	0 m
maximum:	15.1 m

Thickness of the overburden on the west side of the property is:

average:	6.5 m (σ 3.50)
minimum:	1.8 m
maximum:	28.2 m

The thickest part of the overburden is restricted to a small area to the north-east of the West open pit and immediately west of the East open pit and corresponds to a zone straddling the boundary between the Noranda and the Quebec Cartier grids.

3.2 Petrology

Copper mineralization is enclosed within light grey lustrous quartz-sericite phyllites, with lesser amounts of green chloritic phyllite. Light grey sericitic quartzites and dark grey carbonaceous phyllites were also observed in drill logs. Quartzo-feldspathic phyllites, also recorded on the property, are generally barren.

The stratigraphic relation of the phyllites is unknown. They commonly occur interstratified as lenses and discontinuous layers with gradational contacts. The different varieties of phyllites may be repeated several times in the same drill hole showing no systematic vertical variation. Quartzite occurs as thin beds interstratified with phyllite with a very irregular distribution. For all the above reasons, the lithology of the mineralized zones was not represented on the cross sections accompanying this report.

Carbonates are generally associated with swarms of fine veins and fractures irregularly distributed in the rocks. The carbonate content of the copper-mineralized rocks (predominantly dolomite) varies largely and irregularly between 0 and 60%. A rough estimation of the average carbonate content in the mineralized zones and waste to be removed from the proposed open pits is about 12%.

The host rocks for the Hail Harper Creek copper prospect have undergone low-grade regional metamorphism with the development of greenschist facies mineral assemblages.

Metamorphic minerals present within the phyllites include quartz, albite, sericite, chlorite, sphene, carbonate, epidote, tremolite and actinolite.

Deformation which accompanied regional metamorphism was characterized by the development of a strong foliation which transposed the original bedding.

Hydrothermal alteration assemblages have been superimposed on the previous metamorphic assemblages with the development of abundant chlorite which appears to be genetically related to the copper mineralization.

3.3 Mineralization

Pyrite is the most abundant sulfide in the area. In general, the pyrite content varies between 0 and slightly more than 10%. Barren zones generally contain 0 to 2% pyrite, with an average of about 1% pyrite. However, local lens-shaped massive pyrite bodies barren in copper may occur containing up to 50% sulphides.

In the mineralized zones, copper is associated with rocks containing 2 to 10% pyrite (average 5% pyrite). High grade copper is commonly associated with a pyrite content of 4 to 10% or more. In general, while the distribution of copper in the pyrite bodies is rather erratic, the presence of pyrite is a necessary condition for copper values. However, the reciprocal is not the case: high amounts of pyrite may be completely devoid of economical copper (trace of Cu or less than .05 copper).

Chalcopyrite very commonly occurs as a disseminated mineralization along the cleavage planes. It is also observed in tension gashes, in fractures and in quartz veins. The presence of chalcopyrite was also reported in association with pyrite and pyrrhotite in massive sulfide bands and as replacements of pyrite.

Pyrrhotite is also common and frequently observed in the massive sulfide bands.

Other primary metallic minerals of lesser importance include sphalerite, galena, arsenopyrite and molybdenite. Tetrahedrite and bornite are trace minerals. There is no apparent consistent spatial relationship between copper mineralization and any of the above metallic minerals.

Abundant sphene occurs as disseminations or in clusters within chloritic phyllite. Coarse crystallization of rutile was observed as sporadic concentrations in veins and recrystallized zones.

Traces of gold detected were possibly associated with the copper mineralization.

Copper mineralization is confined to masses of lens-shaped pyritic zones that become progressively more tabular-shaped at depth to the north-west part of the property. In the East Zone of the Hail Harper Creek property, the ore-body was probably less eroded and shows copper mineralization mainly confined to more massive bodies of irregularly disseminated pyrite. Below the 1500 metre level, copper mineralization is tabular, striking east-west and dipping moderately to the north with a copper grade which seemingly increases progressively with depth. In the West Zone, the ore is located closer to the surface and the mineralization is principally of the tabular-shaped type. This situation makes the West Zone of the property more suitable for open pit design.

In the East Zone, as well as in the West Zone, the mineralization down the dip towards the north has not been determined.

In the cross sections accompanying the report, the copper mineralization was outlined for different cut-off grades. Ore above 0.2% copper is represented in dashed lines with the intent to show the overall outline of the mineralized zone. Ore above 0.4% copper is represented in plain lines. Dotted lines represent the mineralization above 0.2% copper extrapolated from adjacent sections.

The mineralization plans show the grade distribution in the ore zones at different levels. The areas outlined in these plans correspond to the following cut-off grades:

above 0.2% copper
above 0.3% copper
above 0.4% copper

3.4 Structural Geology

Several periods of deformation can be deduced from the rock descriptions. The earliest recognizable deformation resulted in a foliation subparallel to bedding. A second deformation is indicated by the presence of small scale recumbent isoclinal folds with sheared limbs making the correlation between individual lithological units difficult. Tension fractures may have been generated during this second period of deformation. Non-penetrative shearing associated with slip-folds may represent the youngest period of deformation.

A broad warping of the ore zones with north trending axes could be related to the later stages of deformation. Tension fractures probably related to this warping contain abundant chalcopyrite, pyrite and chlorite.

3.5 Ore Model

The Hail Harper Creek copper prospect was at first generally regarded as being metamorphic/hydrothermal in origin. Recently, the possibility of a volcanogenic-exhalative related mineralization has been accepted more widely. The recent theory is supported by the pyroclastic nature of some of the lithological units, the numerous massive sulfide lenses observed in the area and the likelihood of a mineralization-hydrothermal alteration relationship.

In the context of this new model, the tabular-shaped copper mineralization could be regarded as a possible alteration pipe with copper values apparently increasing in grade and consistency towards the north of the property.

This hypothesis opens up new possibilities and increases the exploration potential of the area. Extending exploration to the north of the property could then result in developing more tonnage with potentially higher copper values possibly associated with precious metals and, ultimately, lead to the discovery of polymetallic massive mineralization.

3.6 Geological Ore Reserves and Grades

Geological Ore Reserves Calculations

Several zones of mineralization have been found on the property. On the East side of the Hail Harper Creek Property (Quebec Cartier), the northern mineralized zone extends from section 420 E to section 450 W, between longitudinal sections 200 N and 100 S. The southern zone is smaller in size and shows lower copper grades. It extends from section 60 W to section 450 W, between longitudinal sections 600 S and 800 S.

On the West side of the Hail Harper Creek property (Noranda), the main mineralized zone which extends between section 480 W and section 1740 W breaks up into a north zone between longitudinal sections 600 N and 200 N, and a central zone between longitudinal sections 100 N and 200 S.

The south zone in the Quebec Cartier side extends into the Noranda grid and consists of at least three narrow bands of copper mineralization, outcropping in an area outlined by cross sections 600 W and 960 W and longitudinal sections 400 S and 900 S.

In the East Zone (Quebec Cartier) as well as in the West Zone (Noranda), an open pit was designed to recover as much ore as possible from the north zone.

Geological ore reserves down to 1500 m Level

assumed specific gravity = 2.6 t/m^3

Quebec Cartier Property (E. Zone)

	Reserves metric tonnes	Grades % Cu
North zone (involved in present open pit)	42,500,000	.39 (cut-off grade .3)
South zone	20,500,000	.30 (cut-off grade .2)
Total E. Zone	63,000,000	.36

Noranda Property (W. Zone)

North zone (involved in present open pit)	54,100,100	.36 (cut-off grade .3)
	or 37,600,000	.38 (cut-off grade .35)
	or 28,200,000	.43 (cut-off grade .35; between sections 870W and 1380W)

(tonnage obtained for different cut-off grades)

Central zone	21,300,000	.42 (cut-off grade .3)
South zone	4,000,000	.40 (cut-off grade .35)
Total W. Zone	53,500,000	.42

Total Reserves

Geological ore reserves down to 1500 m level

Total West and East Zones

West Zone	53,500,000	.42
East Zone	63,000,000	.36
	116,500,000	.39

Total involved in present open pit

W. Zone (north zone)	28,200,000	.43
E. Zone (north zone)	42,500,000	.39
	70,700,000	.41

4.0 MINE PLANNING

4.1 General Description

Two open pits were designed. One is located in the West Ore Zone on the Noranda property and the other is located in the East Ore Zone, on the Quebec Cartier property.

The longer axis of both pits have an east-west direction and are situated approximately along the same line.

The distance between final pit limits on the surface is approximately 500 metres.

The size of the West pit is approximately 1050 x 400 metres (on the original surface) with an area of 422,200 square metres. Average thickness of the overburden is 6.5 metres. Total volume of overburden is calculated to be 2,745,000 cubic metres.

The orebody is generally of a tabular shape and is dipping northwards at an angle of 25 degrees, outcropping along the southern side of the pit. Total tonnage of the pit is 58.1 million tonnes; 24.6 million tonnes of waste and 33.5 million tonnes of ore. There are 15.67 million tonnes of ore above .4% Cu grade, and 7.47 million tonnes between 3 to .4% Cu. Total reserves of ore above .3% Cu grade amount to 23.14 million tonnes with an overall stripping ratio of 1.5:1. The remaining ore grading between .2% Cu and .3% Cu has a tonnage of 10.3 million tonnes.

The best part of the orebody is situated between sections 1080W and 780W, linear distance of approximately 300 metres.

The final pit depth is in the order of 100 metres below surface.

The size of the East pit (on the original surface) is approximately 870 x 580 metres with an area of 519,200 square metres. Average thickness of overburden is 4.6 metres yielding a total volume of 2,388,320 cubic metres. The orebody is of massive shape and becomes tabular with depth. Total tonnage including ore and waste is 121.0 million tonnes, of which 75.8 million tonnes is ore and 45.2 million tonnes is waste. There are 20.7 million tonnes of ore at grade of above .4% Cu and 21.5 million tonnes of ore at a grade between .3 to .4% Cu.

Total reserves of ore grading above .3% Cu are 42.2 million tonnes, with overall stripping ratio of 1.9:1. The remaining ore of a grade .2 to .3% Cu has a tonnage of 33.5 million tonnes. The best part of the orebody is situated between sections 210W to 120E, a linear distance of approximately 300 metres. Final pit bottom is approximately 170 metres below the surface.

4.2 Approach

The general approach was:

- to establish costs of mining and processing the ore at a rate of 4.4 million tonnes annually

- to perform preliminary cash flow analysis over a 10 year period

- to calculate cut-off stripping ratio on a basis of this cash flow analysis

- to draw vertical cross-sections of orebodies along the main axis

- to estimate the economic pit bottom for both open pits using the linear method

- to design open pits within limits imposed by the cut-off stripping ratio

- to estimate mineable reserves in the pits, for Cu grades between .2% and .3%; between .3 & .4% and above .4% Cu

- to estimate equipment in terms of numbers and sizes

- to estimate capital and operating costs of the equipment

- to estimate required manpower

4.3 Cut-Off Stripping Ratio Derivation

The economic depth of a pit is the depth below which ore can no longer be mined at a profit. The first step is to determine the ore/waste ratio where no profit is realized. This is called the cut-off stripping ratio and is equal to:

$$\frac{\text{Value of Ore/Tonne - Average Mining Cost/Tonne}}{\text{Average Stripping Cost/m}^3}$$

(where Value of Ore is sale price minus transportation costs).

To derive the cut-off stripping ratio the following assumptions were made.

- there are 2 open pits
- total tonnage ore: 43,966,000 t
  total tonnage waste: 57,155,800 t
- spec. gravity of ore 2.6 t/m^3
- spec. gravity of waste 2 5 t/m^3
- Project life 10 years
- operating schedule: 350 days/year; 3 shifts/day
- mining equipment: new and used

Metal values

Copper	$3.50/kg
Gold	$20.0/g
Silver	$0.30/g

Capital and operating costs were derived from equipment dealers, reports on mines available in the public domain, and industry practice from privileged information.

The above cost data was input to a computer program to perform cash flow analysis.

From the program output the sum of costs for respective items over the life of mine were calculated. Next, mining costs were divided by total tonnage of ore and waste. This gave average mining and stripping cost per tonne, mill costs, administration, depreciation, royalty taxes and plant services and sales revenue.

These were divided by total ore tonnage to obtain unit costs per tonne of ore. The difference between unit sales cost and operating costs equals the amount of money available for the stripping required to mine one tonne of ore. By dividing this difference by average stripping cost per cubic metre of waste the cut-off stripping ratio of 1.67:1 was obtained. This means that to mine 1 cubic metre of ore the maximum amount of allowable waste to be excavated was 1.67 cubic metres.

4.4 Assumptions Used for Preliminary Pit Design

Vertical cross sections were drawn for each orebody at average intervals of 60 metres.

- Calculated stripping ratio was 1.67:1.
- Pit slopes were assumed to be 45° with no regard for rock type, benches, haul roads, etc.
- By using the linear method an economic pit bottom and pit walls were established graphically for each cross-section of orebody.
- Pit bottoms of all cross sections were adjusted and smoothed to reflect equipment selected and practical considerations as to its operating limitations.
- For each vertical cross section areas of ore, waste and overburden were calculated.

On each section areas of ore grade between .2% Cu and .3% Cu, between .3% Cu and .4% Cu and above .4% Cu were established using weighted averages; these areas were calculated and tabulated.

Ore and waste volumes within each pit were calculated by multiplying areas of ore and waste for each cross section projected to half the distance to adjacent sections on both sides.

Tonnages of ore and waste were calculated by multiplying the volumes of ore and waste by 2.6 tonnes/cubic metre and 2.5 tonnes/cubic metre respectively.

4.5 Basis of Equipment Selection

4.5.1 Assumptions

Equipment selection was done on the basis of annual ore and waste production, averaged from both open pits.

The total annual production is 11,880,000 tonnes of which 4,400,000 tonnes are ore and 7,480,000 tonnes are waste.

At an average ore grade above .3% Cu it was assumed the operating schedule was 350 working days per year, 3-8 hour shifts daily.

Annual operating hours were calculated under the following assumptions:

Overall equipment availability - 75%; utilization of available units 85%; fixed delays were assumed to be 1 hour per shift, i.e. travel, lunch and coffee break.

Using the formula:

working days x shifts/day x % availability
x % utilization of available units
x (8 hours/shift - fixed delays)

the number of operating hours is 4,685 per year.

Shovel and truck sizes were assumed on a basis of the copper mining experience of the general area.

A 9 cubic metre dipper capacity shovel and a 91 tonne truck were selected.

Assuming a 100% fill factor for the shovel dipper and truck box, shovel swing time of 0.5 minutes and truck spot time of 0.3 minutes, the shovel productivity is 1800 tonnes/hour.

The required number of operating shovels is 2.

Truck fleet size was estimated for each pit separately by estimating average haul distances for ore and waste when the pit reached half of its final depth. Average depth for the ore is 80 metres for both pits, whereas for waste it is 30 and 50 metres for West and East pits respectively.

Route profiles for waste and ore transport on surface were established from the topographical plan. It was assumed that maximum road grade within the pit was 8% and maximum grade on surface was 10%.

Average truck speeds on different grades were established from truck performance cards.

Fixed time of the truck cycle was assumed to be 4.3 minutes which includes spot and load time at the shovel and dump time.

The following results were obtained:

West Pit

Ore is hauled 1000 metres at 8% grade uphill in the pit then 450 metres downhill on surface at 0% grade to the crusher. Calculated cycle time is 15.3 minutes. Waste is hauled 375 metres at 8% grade uphill in the pit then 300 metres on +10% grade and 1000 metres at 0%. Calculated cycle time is 15.5 minutes.

East Pit

Ore is hauled 1000 metres on +8% grade within the pit and an additional 200 metres on +8% grade on the surface to the crusher. Cycle time is 14.1 minutes. Waste is hauled 625 metres on +8% grade to the surface, then 300 metres on +10% grade and 1,000 metres at 0%. Cycle time is 17.15 minutes.

Composite cycle time for the truck fleet is 15.87 minutes based on weighted averages.

Required truck fleet size is 8 units assuming average hourly productivity of 344 tonnes per truck per hour.

It was established that a fleet of 8 operating trucks will be sufficient to meet production requirements.

The drilling equipment fleet was established on basis of the copper mining experience of the general area.

Assuming blasthole diameter at 250 millimetres, an advance of 95.5 metres per shift per unit, and a production of 39.8 tonnes per metre, three drills of the B-E 45R size are required.

Basic auxiliary equipment will consist of 3 dozers, 400 hp each, one dozer 300 hp, one grader, one explosives truck, one maintenance truck and one water truck.

Capital costs of equipment were obtained from equipment dealers and operating costs were obtained from published copper mining data from the general area.

The maintenance and repair workshop will provide 2 bays for haulage trucks, one repair bay for support equipment and one repair bay for service trucks. Areas for tire, lube and welding shops are provided.

The number of employees was established on basis of the number of units of equipment, 3 shifts per day operation and comparison with other mines of relatively similar size.

The lists of equipment, purchase and operating costs and staff list are included.

4.5.2 Calculations

Annual ore production	4,400,000 t
Annual waste production	7,480,000 t
Total production	11,880,000 t

Annual Operating Hours =
working days x shifts/day x % availability
x % utilization of available units x
(8 hours/shift - fixed delays)

350 days x 3 shifts x 0.75 x 0.85 x (8 hours - 1 hour)

= 4,685 hours/year assumed 4,685 hours/year

Daily Production:

Ore $\dfrac{4{,}400{,}000\ \text{t}}{350\ \text{days}}$ = 12,571 t/day

Assumed 12,570 t/day

Waste $\dfrac{7{,}480{,}000\ \text{t}}{350\ \text{days}}$ = 21,371.4 t/day

Assumed 21,370 t/day

Total 33,940 t/day

Assumed 9 m^3 Electric Shovel
and 91 t trucks

Shovel Productivity:

$$\text{t/h} = \frac{60\ \text{m/h}}{\left(\dfrac{\text{truck box m}^3}{\text{shovel dipper m}^3} \times \text{shovel swing time}\right) + \text{Truck Spot Time}} \times \text{Truck Box (t)}$$

Assuming 100% Fill Factor
Truck Box = 48.6 m^3
Shovel Dipper = 9 m^3
Shovel Swing Time: 0.5 min
Truck Spot Time: 0.3 min

Shovel Productivity = 1800 t/h

$$\text{Number of Shovels} = \frac{\text{Ore + Waste Production t/yr}}{\text{Shovel Productivity x h/yr}}$$

$$= 1.40$$

Assumed 2 shovels

Truck Fleet Estimation

Both pits - ore haulage

Average lift in the pit 80 m at 8% grade

Waste haulage: Av. lift in West pit is 30 m @ 8% gr.
Av. lift in East pit is 50 m @ 8% gr.

Ore Trucks - Route Profile

West Pit - Ore Haulage

Haul 1000 m @ 8% grade + 450 m @ -10% grade
Return: 450 m @ + 10% grade + 1,000 m @ -8% grade

Truck cycle time: Haul time + return time
+ load time + spot + dump time

$$\text{Load Time} = \frac{\text{Truck Box m}^3}{\text{Shovel Dipper m}^3} \times \text{shovel swing time}$$

$$= \frac{48.6\ \text{m}^3}{9\ \text{m}^3} \times 0.5\ \text{min} = 2.7\ \text{min}$$

Spot Time = 0.3 min

Turn & Dump Time = 1.3 min

Total Fixed Time = 4.3 min

Truck Speed:

Loaded	on + 8% Grade	= 11 km/h
	on level	= 22 km/h
	on -10% grade	= 22 km/h

Empty speed	@ 10% grade	17 km/h
	@ 8% grade	22 km/h

Ore Cycle Time 15.3 min

Waste Truck

Haul 375 m on +8% grade + 300 m on +10% + 1000 m on level

Return 1000 m level + 300 m on -10% + 375 m on -8%

Speed: Loaded on +10% = 9.33 km/hr

Waste Cycle time: 15.5 min

<u>East Pit</u>

Ore Truck

Haul 1000 m on + 8% + 200 m on + 8%

Return: 1200 m on -8%

Ore Cycle time: 14.1 Min

Waste Truck

Haul 625 m on +8% + 150 m level + 300 m on +10% + 1000 m level

Return: 1000 m level + 300 m on -10% + 150 m level + 625 m on -8% grade

Waste Cycle time: 17.15 min

Composite cycle time

$$\frac{\text{Cycle time x t}}{\text{t}}$$

For West Pit

$$\frac{(15.3 \text{ min x } 12{,}570 \text{ t}) + (15.5 \text{ min x } 18{,}855 \text{ t})}{31\ 425 \text{ t}}$$

Composite cycle time is 15.42 min

For East Pit

$$\frac{(14.1 \text{ min x } 12{,}570) + (17.15 \text{ min x } 23{,}883 \text{ t})}{36{,}453 \text{ t}}$$

Composite cycle time is 16.10 min

Truck Fleet Unit Required

$$\frac{\text{Annual t}}{\text{Annual oper. h x t/h/unit}}$$

Truck Productivity per hour (longest cycle time = 16.10 min).

$$\frac{60 \text{ min}}{16.10 \text{ min}} \quad \text{x } 91 \text{ t} = 339 \text{ t/h}$$

$$\frac{11{,}880{,}000 \text{ t}}{4685 \text{ hrs x } 339 \text{ t/h}} \quad = 7.48 = 8 \text{ trucks}$$

Drilling Equipment

(B-E 45R Drill 250 mm dia.)

95.5 m/shift x 39.8 t/m = 3,799 t/shift, assumed 3,800 t/shift

3,800 t x 3 shifts = 11,400 t/day

Daily tonnage 33,940 t/day

Number of drills = $\frac{33{,}940 \text{ t/day}}{11{,}400 \text{ t/day}}$ = 2.98, assumed 3 drills

5.0 PROCESSING PLANT

5.1 General Description

The process selected for the 4.4 million t/a milling process rate requirement consists of a 1370 mm (54-inch) gyratory for primary crushing followed by conventional crushing and grinding (rod mill ball mill system), flotation concentration and dewatering.

For the purposes of the preliminary feasibility evaluation the alternative of autogenous or semi-autogenous primary grinding has not been evaluated in detail. This would require laboratory testing to assess the competence of this relatively soft ore to form adequate media. On a preliminary basis this approach would be expected to reduce mill capital by about 5%, while increasing operating costs by 2 to 5 percent.

All process design is based on preliminary testwork carried out by Noranda Mines Ltd., as summarized in the Noranda Ore Dressing Laboratory report titled "Preliminary Flotation Testwork on Harper Creek Property Ore Samples" dated April 27, 1971.

5.2 Process Design Criteria

The preliminary flowsheets were designed to conform to the following guidelines:

Estimated average grade of mineable ore	
Mining, ore and waste	3 shifts, 7 days or 20 operating shifts per week

Primary crushing	10 hours per day, 7 days per week
Secondary and tertiary crushing	3 shifts, 7 days/week chosen with provision for estimated maintenance
Milling rate	4,400,000 t/a, 350 days = 12,600 t/d
Milling recovery	87% Cu minimum
Concentrate grade	28 Cu, minimum (dry basis)
Ratio of concentration	87:1

5.3 Performance Specifications

5.3.1 Capacity

5.3.1.1 Daily Capacity of Crushing Plants

The plant will be required to handle 12,600 dry t/d of open-pit ore. The ore will be assumed to contain about 5% moisture.

5.3.1.2 Availability of Crushing Plants

.1 Primary Crushing

Routine ore production will average 10 hours per day. Primary crushing will be operated to accommodate this schedule.

.2 Secondary and Tertiary Crushing

Secondary and tertiary crushing will be provided for three shifts. Crusher availability is based on allowing 2 hours per day and one 8-hour shift per week for preventative maintenance. This will provide an average of 20.8 available hours per day or 87% of total time. To allow for contingencies, actual operating time is taken as 95% of available time, or 19.8 hours per day (82.5%).

5.3.1.3 Hourly Crushing Rate

.1 Primary Crushing

For Scheme No. 1, the dumping cycle at the primary crushing station is estimated at 75 t every 3.5 minutes over a continuous 10-hour period. The average feed rate to the primary crusher will be 1,260 t/h.

A 1,370 mm (54-inch) gyratory crusher is rated at up to 1,600 t/h capacity.

.2 Secondary and Tertiary Crushing

The design capacity based on crushing for three shifts with about 83% overall availability will be 630 dry t/h.

The capacity of a 2130 mm (7 feet) standard secondary crusher ranges from 800-900 t/h depending on the discharge setting.

A single tertiary crusher set at 16 mm (5/8 inch) equipped with an extra coarse bowl, has an approximate total capacity (net finished product plus circulating load) of 500 t/h, or throughput capacity in closed circuit of 325 t/h. Two are required.

5.3.1.4 Daily Capacity of Grinding Plant

The grinding plant must grind 12,600 dry t/h.

5.3.1.5 Availability of Grinding Plant

Availability of the conventional grinding circuit is based on operating 95% of the time. This provides for an average of 8-hours downtime for each mill each week.

5.3.1.6 Hourly Grinding Rate

The design capacity based on 95% availability is 550 t/h for the rod mill and 275 t/h for each ball mill circuit.

5.3.2 Description of Ore and Products

5.3.2.1 Raw Ore to Crushing Plants

The feed to the crushing plant will consist of open pit material with a maximum lump size estimated to be 1200 x 1200 mm. Surface moisture is estimated to be 5%.

5.3.2.2 Crushing Plant Product, Rod Mill--Ball Mill Feed

The gyratory crusher will provide a minus 150 mm (6-inch) product, of which about 15% will be minus 19 mm (3/4-inch). The minus 19 mm (3/4-inch) material bypasses the fine crushing plant, being conveyed directly to the mill feed bins. The 150 x 19 mm (6- x 3/4-inch) material is directed to a stockpile for reclaiming to the secondary crusher. Removal of the fine material from the ore stream will lessen freezing hazards in the stockpile. Ball mills will follow the rod mills to produce the objective grind.

5.3.2.3 Grinding Plant Product

The grinding plant is designed to produce a flotation feed product containing about 62% minus 200 mesh.

5.3.3 Description of Crushing and Grinding Facilities

(See Drawings Nos. 88005-001 and 88005-GA-001)

5.3.3.1 Primary Crushing Plant

Open-pit ore will be delivered to the surface crushing plant at the rate of 12,600 t/d via 91 t trucks. Blasting is expected to yield a maximum lump size measuring 1200 x 1200 x 1200 mm (4 x 4 x 4 feet). Trucks will be able to dump from one side. The primary crusher will be a minimum of 1,000 m from the open pit. Trucks will dump directly into the 1370 mm (54-inch) gyratory crusher with an Open Side Setting (O.S.S.) of 150 mm (6 inches).

The crusher product will fall into a 250 t surge hopper, equipped with one 1500 x 8000 mm (60-inch wide, 26-foot long) pan feeder which will discharge to a 1370 mm (54-inch) belt conveyor. All materials-handling equipment following the primary crusher will be designed to accommodate the rated capacity of the crusher at 1600 t/h.

A dust control exhaust system, rated at 100,000 m^3/h will be included in the crusher building.

The primary crusher product will be delivered via a 1370 mm (54-inch) conveyor to one extra-heavy-duty 2440 x 7000 mm (8- x 20-foot) double-deck primary screen. The top deck will be stepped manganese rails with 50 mm (2-inch) openings. The bottom deck will have 19 mm (3/4-inch) slotted or rectangular openings. The primary screen undersize will be transported via a system of 700 and 1070 mm (30 and 42-inch) conveyors, a 1220 mm (48-inch) tripper conveyor and tripper to four fine-ore bins having a total live capacity of 10,000 t which allows about 18 hours of grinding operation. To prevent freezing of the fines during the winter, a hot air heating arrangement will be included.

Screen oversize will be transported via 1220 mm (48-inch) belt conveyor, equipped with a weightometer for inventory, to a coarse-ore stockpile. Total storage will be 25,000 t which will provide material for about 2 days of operating time.

5.3.3.2 Secondary Crushing

Four 914 x 1524 mm (36- x 60-inch) mechanical-type vibrating feeders will reclaim and feed coarse ore via a 1070 mm (42-inch) belt conveyor, equipped with a tramp iron detector and magnet, to one secondary crusher with Closed Side Setting (C.S.S.) of 45 mm (1-3/4 inches).

Crusher capacity at this setting ranges from 800 to 900 t/h.

Secondary crusher discharge will be sent via a 1220 mm (48-inch) conveyor to a feed bin which will split feed to two heavy-duty 2440 x 6100 mm (8- x 20-foot) double-deck secondary screens. The top deck will have 38 mm (1-1/2-inch) openings; the bottom deck will have 19 mm (3/4-inch) slotted openings.

5.3.3.3 Tertiary Crushing

Two tertiary crushers with C.S.S. of 16 mm (5/8-inch) will operate in closed circuit with the two 2440 x 6100 mm (8- x 20-foot) double-deck secondary screens. Product from the tertiary crushers will discharge and join secondary-crusher discharge on a common 1220 mm (48-inch) conveyor and a pair of 1220 mm (48-inch) scissor conveyors. The material will discharge into the feed bin and thereafter be fed to the secondary screens, via two 1520 x 2450 mm, (60- x 96-inch) mechanical-type vibrating feeders. Each feeder is rated at 1100 t/h.

Primary- and secondary-screen undersize, minus 19 mm (3/4 inch) will join and then pass through a system incorporating 105 mm (42-inch) conveyor, a 1220 mm (48-inch) tripper conveyor, weightometers and samplers. The 1220 mm (48-inch) tripper conveyor and a tripper will transport the material to four fine-ore bins as described previously.

Dust-control equipment will be provided at all major dust points in the secondary- and tertiary-crushing circuits, as well as in the screening and conveying systems. The exhaust system is rated at 177,000 m^3/h which includes the requirements for coarse-ore reclaim. For the fine-ore bins, 62,000 m^3/h will be provided.

5.3.3.4 Grinding

Slot feeders, 3660 mm (12-feet) long by 250 to 660 mm (10- to 26-inch) wide taper, will discharge minus 19 mm (3/4 inch) feed from the fine ore bins via a 1220 mm (48-inch) mill-feed conveyor equipped with a weightometer and sampler, at 550 t/h to the grinding circuit. The grinding circuit consists of a 4000 ∅ x 6100 mm (13 ∅x 21 foot) rod mill and two 4900 ∅ x 4900 mm (16 ∅x 16 foot) ball mills in parallel.

The rod mill will be fed with a tube feeder integrated with the mill-feed conveyor. Conveyor speed will be regulated by a belt scale which will weigh, record, and control a preset feed rate to the mill. Discharge from the rod mill is divided, 50 percent to join each ball mill discharge in a common pump box from which the pulp will be pumped via two 355 x 305 mm (14- x 12-inch) pumps, to four 760 mm (30-inch) cyclones in closed circuit with the ball mill. The primary cyclones will be operated at 35 kPa (5 psi).

For each ball mill, an extra cyclone will be included as standby. There will be standby pumps.

A bond work-index value of 10.0 (metric) was used to estimate the horsepower requirements of the mills.

Cyclone underflow, at 70% solids by weight, will discharge directly into the ball mill. Ball mill discharge at 65% solids, will be sent to a pump box to join rod mill discharge for feed to the cyclones. Cyclone overflow, at 37% solids (range 35 to 40% solids) will be pumped with two 355 x 305 mm (14- x 12-inch) pumps to the flotation circuit.

Drawing No. 88005-001 is the flowsheet for the conventional grinding circuit showing the circuit material balance.

A small ball mill slaker handling about 3 t/d of lump bulk quicklime, will be required to prepare milk-of-lime slurry for flotation. Alternatively a slurry of finely-ground hydrated lime may be used. Tanks and piping will be required to store, prepare and feed milk-of-lime.

Since the service is abrasive, rubber linings will be used to reduce wear in pumps and cyclones throughout the circuits.

5.3.4 Flotation

(See Drawings 88005-002 and 88005-GA-002)

Drawing 88005-002 is a flowsheet of the flotation process and material balance.

5.3.4.1 General

The routine of products and flows within the flotation section is tentative. In the cleaner flotation stages, all flotation cells will be equipped with double launders to allow operators to re-route flotation products as required. Product routing will depend on such factors as:

- Liberation achieved during regrinding
- Assays of flotation products
- Recovery of slimes
- Recovery of flotation agents
- Excessive accumulation of flotation reagents in circulating flows

Future laboratory testing will firm up these variables.

5.3.4.2 Rougher Flotation Circuit

The cyclone overflow from each ball mill circuit will be pumped with two 355 x 305 mm (14- x 12-inch) pumps to a 3050 mm (10-foot) diameter by 3666 mm (12-foot) deep conditioner. Pulp density will be 37% solids, but is expected to vary from 35 to 40% solids. Each conditioner will provide 2.5 minutes of conditioning time. R343 (sodium isopropyl xanthate) promoter and MIBC frother are added at 0.05 and 0.015 kg/t of feed respectively.

Each conditioner will be equipped with automatic pulp samplers. From each conditioner, the pulp will be delivered to individual banks of rougher flotation cells. Each rougher flotation section will have twelve 15 m^3 (500-cubic foot) No. 120A Agitair machines. Retention time for rougher flotation will be about 20 minutes.

Rougher flotation is expected to produce a concentrate at a ratio of concentration of 18 to 1, or about 5.5% of the flotation feed. Rougher concentrate grade is expected to be 7.5% Cu.

Rougher flotation tailings will be sampled and pumped via two 355 x 3.5 mm (14- x 12 inch) pumps to the tailings dam.

5.3.4.3 No. 1 Regrind Circuit - Regrinding Rougher Flotation Concentrate

The rougher flotation concentrate will be collected in a sump and pumped to two 30 cm (12-inch) diameter cyclones of the regrind circuit. There will also be a standby cyclone and pump. The cyclones will be operated at 80 kPa (11 to 12 psi) and will be in closed circuit with a 1500 x 3050 mm (5- x 10-foot), 75 kW (100 HP) grinding mill.

Grinding media will be steel balls.

Mill discharge will be collected in a sump to join rougher flotation concentrate as combined feed to the cyclones. Circulating load will be 250% of new feed. If required, consideration will be given to open-circuit regrinding. Other alternatives include using mechanical classifiers or Dutch State Mines (D.S.M.) curved screens instead of cyclones.

5.3.4.4 First Cleaner Flotation

Regrind cyclone overflow at 25% solids, will pass through an automatic sampler and will be collected in a sump and pumped to the first cleaner flotation section. 0.005 kg/t of S3302 promoter alyl amyl xanthate ester will be added, along with 0.01 kg/t of sodium cyanide and 0.25 kg/t of lime (as 20% solids slurry).

The first cleaner flotation section will consist of six 2.8 m^3 (100-cubic foot) No. 60 Agitair machines which will provide about 9 minutes retention time. Tailings will pass to an identical first cleaner scavenger flotation section of an additional six 2.8 m^3 cells.

Concentrate from first cleaner flotation will be sent to second cleaner flotation regrinding, whereas scavenger concentrate will be returned to regrinding.

5.3.4.5 Cleaner Scavenger Flotation

The cleaner scavenger tailing is expected to contain too high a concentration of sodium-cyanide reagent to allow its direct internal reclaim. It will be collected in a sump, passed through a magnetic flowmeter, and pumped to join the tailings stream. There will be provision for sampling the cleaner scavenger tailings at the sump.

5.3.4.6 Second Cleaner Flotation

The first cleaner concentrate will be collected in a sump and then pumped to the second cleaner flotation cells. There will be five No. 21 Denver "Sub A" flotation cells each having 1.1 m^3 (40-cubic feet) of volume. Retention time will be about 13 minutes. The second cleaner tailings will be collected in a sump pumped to join the reground rougher flotation concentrate and will be fed to first cleaner flotation.

5.3.4.7 Reagents and Reagent Feeding

For pH adjustments, lime will be added as 20% solids slurry of milk-of-lime, which can be prepared in any of the following ways:

- Grind bulk quicklime (plus 1/4 to minus 1-inch pebbles) in a ball mill in closed circuit with a small classifier. This circuit will treat about 9 tons of bulk lime per day. Pump the classifier overflow, at 20% solids, to a lime storage tank.

- Slurry pulverized quicklime or slaked lime at 20% solids in a mixing tank. Pump the slurry to the lime storage tank in the concentrator.

- Slake quicklime in a suitable slaker capable of handling quicklime in pellet form.

R343 and S3302 promoters are fed as 20% aqueous solution. MIBC is fed as received.

Sodium cyanide will be prepared as a 10% aqueous solution, as a pyrite depressant.

The xanthates, sodium cyanide, and lime will be prepared in mixing tanks equipped with suitable mixing motors. Prepared reagent solutions will be pumped to their respective storage tanks.

MIBC used at full strength will be kept in storage tanks.

Tank sizes are based on reagent requirements for 24 hours of continuous operation.

Lime slurry will be continuously circulated in a loopline to prevent plugging by settled solids. Process addition will be by pH control.

All other reagents will be pumped to their delivery points by diaphragm type variable speed metering pumps.

5.3.4.8 Dewatering of Final Copper Concentrate

.1 Thickening

The final copper concentrate will be collected in a sump and pumped to a 5 m (16 foot) diameter high capacity thickener. To aid settling, a flocculant, such as Separan (American Cyanamid Co) will be added at an estimated dosage of 13 g/t (0.03 pounds per ton) of concentrate, added as a 0.05% strength solution.

Thickener overflow at 5 m^3/h (22 US gpm) will be sampled, and passed through a magnetic flowmeter, before return to the grinding circuit.

The thickened final copper concentrate, at 50 to 60% solids, will be discharged into a 3000 Øx 3200 mm (10 Ø x 10.5 feet) concentrate surge tank and then pumped to filtration.

.2 Filtering

The final concentrate will be pumped to a 1830 mm (6-foot) diameter, 5-disc filter having a capacity of 250 t/d. The filtrate at 1.6 m^3/h (7 US gpm) will be returned to the final concentrate thickener. The filter will have suitable vacuum and air-blow equipment, as well as provision to recirculate boot overflow to the concentrate surge tank.

The moisture content of the filter cake is expected to be 15 to 20%.

5.3.4.9 Tailings Disposal

.1 Including Tailings Thickeners

At the definitive feasibility stage a cost benefit analysis will be required to evaluate the need for tailings thickening. If tailings thickeners are required, tailings from each rougher flotation section along with first cleaner scavenger tailings will be collected in their respective sumps, pass through automatic samplers, and be pumped to two 80 m (260-foot) diameter by 6.7 m (22-foot) centre-depth tailings thickeners. Thickener underflows, at 50-60% solids will be pumped to the tailings pond.

.2 No Tailings Thickeners

If tailings thickeners are excluded, as assumed for the present study, rougher flotation tailings will pass through automatic samplers and will join in a common collection box. First cleaner scavenger tailing will also join rougher flotation tailings at the collection box, which will provide a convenient sampling point for the combined waste products.

The combined tailings, at 36% solids, will be discharged to the tailings pond at a rate of 1125 m^3/h (4950 US gpm) via a 610 mm (24 inch ID) sclair pipe tailings line to the pond.

5.3.4.10 <u>Water Supply</u>

.1 Including Tailings Thickeners

If included, overflow from both tailings thickeners will be reclaimed for use in the milling circuit. Total available overflow from thickening will be 500 m^3/h (2200 US gpm). Reclaimed overflow will be pumped through a 406 mm (16-inch) inside diameter pipe to a 13 m (42-1/2-foot) diameter by 15 m (48-foot) high process water tank. Storage capacity will be 1925 m^3 (500,000 US gallons), which represents 2 hours operation at the maximum calculated rates.

The total amount of water required for milling will be 910 m^3/h (4,000 US gpm). Reclaimed water, which will be recycled within the milling circuit, will include: overflow from the concentrate thickener 23 m^3/h (100 US gpm), overflow from tailings thickeners 500 m^3/h (2200 US gpm).

Total reclaimable water will be 523 m^3/h (2300 US gpm) to which 387 m^3/h (1702 US gpm) of new water will be added to meet milling requirements. The additional water will be obtained from a fresh water pond, which will be developed in the reservoir formed at the toe of the tailings dam.

.2 No Tailings Thickeners

Along with the tailings discharged to the tailings pond, there will be a total of 945 m^3/h (4158 US gpm) of water. Assuming that the sands in the tailings pond settle to 70% solids, the maximum amount of recoverable process water will be 713 m^3/h (3140 US gpm).

Since the concentrator will require a total of 910 m^3/h (4000 US gpm) of water, of which 713 m^3 are recoverable, 197 m^3/h (867 US gpm) of fresh water will be required. The fresh water will be available from the water reservoir previously described.

5.3.4.11 Sampling

Automatic samplers will be used on principal products such as the flotation feed, rougher flotation tailings, cleaner scavenger flotation tailings, and overflow from the final concentrate thickener.

All piping that will deliver flows into sumps and pump boxes will be arranged to enable physical ease of sampling of products.

5.3.4.12 Instrumentation and Control

Magnetic flowmeters will be used on cleaner scavenger flotation tailings and overflow from the final concentrate thickener. Pulp level controls will be installed in the flotation machines.

5.3.4.13 Pilot Plant Test Area

There will be a moderate space allowance of 10 x 10 m for a pilot-plant section to treat a portion of rougher flotation tailings in order to evaluate recovery of minor amounts of heavy byproduct minerals. Basic equipment will include four-rougher and two-cleaner Humphreys spiral concentrators.

5.3.4.14 Metallurgical Laboratory

A metallurgical laboratory will be located at the project site. Basic equipment will include grinding mills, flotation machines, filters, drying ovens, balances, and sizing devices. There will be provision for sample preparation, sample storage and dust control.

6.0 INFRASTRUCTURE

6.1 General Description

The mine site is located some 15 kilometres south of the communities of Birch Island and Vavenby on Highway 5, east of Clearwater and south of the North Thompson River as shown on Figure 6.1.

The land south of the community rises steeply from an elevation of under 500 metres to the site elevation of over 1600 metres.

The site can be accessed from Highway 5 through either Birch Island or Vavenby. Local access is presently via an unimproved forest service road known as the Jones Creek Forest Service Road.

A Canadian National Railway line is located on the north side of the river.

The closest paved airport is at Kamloops, over 100 kilometres to the south.

The area is considered by the Ministry of Environment and Parks in Kamloops to be relatively remote with no apparent significant environmental concerns. The Ministry also advised that the North Thompson River carries a great deal of debris during spring runoff and glacial sediment all year round. The flow is fairly low except in spring. They also indicated that there are not a great deal of migrating fish in the river at this point.

The site facilities will be located between the two mine areas or about 850 metres from the centre of each of the pits as shown on

Figures 6.1 and 6.2. The existing site grade across the site facilities area is in the order of 14%.

Overburden from the mine will be used to grade the site area to accommodate the process buildings, maintenance shop and warehouse, administration building and laboratory and the utilities support facilities such as the water storage tanks, diesel tanks and sewage treatment plant.

In addition, overburden from the mine stripping will be used for the construction of access roads such as those to the tailings pond and mine waste dump.

The tailings pond, mine waste dump and settling ponds will be located as shown on Figure 6.1. The tailings pond is approximately 3 kilometres from the process plant, the mine waste, 2 kilometres and the settling ponds 0.5 kilometres and 2.5 kilometres.

Power supply is assumed to be available from the 138 kV line on the north side of the North Thompson River. B.C. Hydro has cautioned that this line is nearing capacity and they foresee the need to possibly impose a capital charge on the development for upgrading of their grid in the general area.

Water is assumed to be available from the North Thompson River for pumping to the site.

6.2 Tailings Ponds

The process plant is expected to present some 35 million cubic metres of tailings for disposal over the projected 10 year mine life.

A number of sites were examined for a tailings pond location. Although the choices are limited, a suitable pond can be developed on one of the branches of Harper Creek about 3 kilometres from the process plant. The 35 million cubic metres can be disposed of behind a dam of approximately 100 metres high and 600 metres long as shown on Figure 6.1. This dam, at its maximum height, will require 6.2 million cubic metres of material.

The proposed top level of the dam will be about 200 metres below the main site facilities. Since this pond is expected to be acid generating, leachate will be collected and pumped back to the process plant for treatment.

It is expected that after startup reclaim water can be pumped back to the process plant at a rate of about 680 cubic metres per hour.

6.3 Waste Water/Settling Ponds and Site Drainage

Surface water drainage in the pit and waste rock dump areas will be intercepted in perimeter ditches and conveyed to settling ponds as shown on Figure 6.1.

Discharge of the clarified water from the ponds will be to Baker Creek.

General site drainage from the site facilities area and drainage from the pit area will also be collected for similar treatment.

6.4 Water Supply

Water will be required for processing, fire protection as well as for a potable supply.

The total quantity of water to be required for the plant is in the order of 1,000 cubic metres per hour. As noted earlier about 680 cubic metres per hour is assumed to be obtainable from the tailings pond after startup. This leaves a balance of about 320 cubic metres per hour to be made up from an external source.

A number of alternatives were considered including pumping from the North Thompson River, and storage constructed on the Harper Creek.

Groundwater supply is not considered to be a feasible alternative for this location.

Preliminary analysis indicates the most favourable alternative to be pumped storage from the North Thompson River.

Pumping 320 cubic metres per hour from the river will require a pump station and a booster pump station each with two 500 hp pumps and about 8 kilometres of 250 millimetre diameter pipeline.

Since the North Thompson is reported to be silty during spring runoff and variable in flow during the summer months, it is likely that a settling basin will be required at the intake.

Three storage tanks will be provided on site for process water (2,300,000 litres) fire protection (1,400,000 litres) and potable water (230,000 litres).

6.5 Power Supply

The mine complex is expected to require a 15,000 kVA transformer. The processing plant will require approximately 10,000 kVA and the remaining mine facilities approximately 7,000 kVA. This will be provided from the B.C. Hydro 138 kV power line on the north side of the North Thompson River and transmitted to the site along a right of way generally paralleling the water main and the access road. A main transformer and 138 kV substation will be located at the site facilities area. Power will be distributed at 13.8 kV to the various facilities around the plant, including the open pit and tailings dam. The main plant itself will be supplied at 4.16 kV via a 7 500 kVA step down transformer. 4.16 kV will also be used as a distribution voltage to peripheral areas of the plant, including the workshop and laboratory buildings.

6.6 Sewage Treatment

Sewage will be collected from the washrooms located in the administration office, and the other main facilities buildings. It will be conveyed to a package aerobic treatment plant where effluent will be discharged to Baker Creek.

For the purposes of this report and for sizing the plant it is assumed that 300 persons will be employed on the site and on this basis the plant will be capable of handling 30 cubic metres per day of sewage.

6.7 Roads

The site is presently accessed by the Jones Creek Forest Service Road. This road has generally been located on the most favourable alignment to access the mine area.

For the purposes of this report it is assumed that this road will be 14 kilometres long, widened to 20 metres, surface gravelled, ditched and the alignment improved. Extra width will be provided on sharp bends.

The existing grade averages around 10% which, although very steep, is assumed to be acceptable for the limited access required.

6.8 Buildings

The principal site buildings are shown on Fig. 6.2.

6.8.1 Process Buildings

The crusher buildings will be structural steel, insulated metal sheathed buildings and will be heated. The mill building will be a structural steel frame with insulated metal sheathing on roof and sides and will house the mills, flotation, drying and packaging. Within the structure will be offices for operating personnel and the metallurgical laboratory. There will also be a change house and sanitary facilities for personnel. Conveyor galleries connecting the mills with the crusher will be covered and heated.

6.8.2 Maintenance Shop and Warehouse

This building will consist of a structural steel frame with insulated metal sheathing for roof and sides and will contain a main gallery for the repair of the 100-ton trucks and other heavy equipment. The other section of the building will consist of a two-storey warehouse and a two-storey office structure with adjacent dry rooms and sanitary facilities. The plant purchasing office will be located in this building adjacent to the warehouse section. Offices for shift foremen, both in service and mine, will be located in the building. Adjacent to this warehouse, there will be a prefabricated metal-sheathed building that will serve as a construction warehouse. It will also contain office space for the construction contractor. It is not intended that this building will be part of the permanently heated plant.

6.8.3 Administration Building and Laboratory

This building will consist of a wood frame with metal insulated sheathing consisting of coloured panels and sun shades and will house the administrative personnel of the project in the main structure and the analytical laboratory and operating personnel for the laboratory, and engineering and surveying personnel in a single storey adjacent "L". The office building and laboratory will have heating and air conditioning units and lunch rooms, and sanitary facilities will be provided for all personnel. A vault at each level will provide storage space for engineering, accounting and laboratory documents.

6.9 Construction Camp

The nearby communities of Clearwater, Vavenby and Birch Island are assumed to be able to provide accommodation during construction. The construction camp will, as a result, be assumed to provide for only daily needs.

6.10 Fire Protection

As indicated in Section 6.4, water will be stored in a reservoir at the main site for fire protection. A distribution system complete with hydrants will be provided.

7.0 COST ESTIMATES

7.1 Operating Costs

7.1.1 Basis for Estimate

Operating costs are based on an average of 12,600 dry tonnes of ore per day for 350 operating days per year, or 4,400,000 dry tonnes.

7.1.2 Freight Costs

Total freight cost applied to steel supplies is as follows:

Trucking costs from Vancouver or Kamloops to the Hail Harper site are charged at $0.25 per tonne kilometre.

7.1.3 Power Consumption and Costs

Power is based on supply from the existing transmission line at 0.029/kWh, as quoted. Power cost is based on B.C. Hydro.

7.1.4 Reagent Consumption and Costs

Reagent consumption and cost data are based on the Noranda Laboratory results. Reagent costs represent vendors' estimates. The vendors have recommended an estimated price escalation of 3% to 5% per year. Reagent consumption and costs are summarized in Table 5.

7.1.5 Steel Consumption and Costs

Steel consumption for crushers, grinding mills and other items such as feeders and screens are based primarily on the average of plant data representing operating experience. Steel costs represent the vendor's estimates. Steel consumption and costs are summarized in Table 7.13.

7.1.6 Heating Costs

In the available time it has not been possible to establish the relative economics of fuel oil heating from the existing pipeline in the North Thompson Valley, the availability of natural gas in the area and electric heating. An allowance of $1,000,000 per year ($0.2273/t) has been made for heating.

7.1.7 Labour Costs

Labour costs are based on the manning tables included herein. Annual working hours per man are estimated at 2,000. Tables 15 to 17 summarize the labour costs.

7.1.8 Supplementary Operating Costs

Milling power costs were calculated on the basis of a "10" work index and 0.0290 per kWh power cost.

7.1.9 Costs Per Tonne

Mining $/tonne values are based on ore plus waste; milling costs per tonne are based ore to process only.

7.1.10 General Overhead Charges Not Included

These are charges, not including labour, to operate and maintain auxiliary facilities and corporate management. These charges include, office supplies, telephone and telegraph, staff travel, lab supplies, medical and safety supplies.

Table 7.1 - Overall Operating Costs Summary

	Mine $	Mill $	H.O.Admin. $	Services $	Total $
Equipment	7,579,425	-			7,579,425
Labour	5,111,600	3,932,500	854,000	1,454,700	11,352,800
Materials	1,785,110	4,742,640			6,527,750
Power	217,772	2,413,568			2,631,340
Heating	-	1,000,000			1,000,000
Total Annual Cost	14,693,907	12,088,708	854,000	1,454,700	29,091,315
$/tonne Ore + Waste	1.2369	-	-	-	
$/tonne Ore only	3.3395	2.7475	0.1941	0.3306	6.6117

Table 7.2 - Total Annual Mine Operating Costs

Item	Cost Per Year	Cost per Tonne Ore + Waste
Equipment	$7,579,425.	$ 0.6380
Labour	5,111,600.	0.4303
Power	217,772.	0.0183
Supplies	1,785,110.	0.1503
TOTAL	14,693,907.	1.2369

Cost Per Tonne = $1.2369 (Ore + Waste)

Cost Per Tonne = $3.3395 (Ore only)

Table 7.3 - Annual Mine Equipment Operating Cost

	No. of Units	Oper. Cost Per Hour	Oper. Hours Per Year	Annual Cost Excluding Wages
Haul Truck	8	$ 75.00	4685	$ 2,811,000.
Shovel	2	150.00	4685	1,405,500.
Drill	3	90.00	4685	1,264,950.
Dozer, D9	3	90.00	4685	1,264,950.
Dozer D8	1	60.00	4685	281,100.
Grader	1	45.00	4685	210,825.
Explosives Truck	1	15.00	2340	35,100.
Maintenance Truck	1	7.50	2340	17,550.
Water Truck	1	75.00	3510	263,250.
Pickup Truck	4	3.00	2100	25,200.
				$ 7,579,425.

Cost Per Tonne: $0.6380 (ore + waste)

Table 7.4 - Total Annual Mine Supervisory and Labour Costs

Classification	Number of Personnel	Base Cost/Year $	Benefits @ 30% $	Total Cost Per Year $	Cost/Tonne Ore + Waste $
General Engineering	7	240,000.	72,000.	312,000.	0.0263
Operating Labour	83	2,490,000.	747,000.	3,237,000.	0.2725
Maintenance Labour	43	1,202,000.	360,600.	1,562,600.	0.1315
	133	3,932,000.	1,179,600.	5,111,600.	0.4303

Table 7.5 - General Mine Engineering Costs

Classification	Number of Personnel	Cost Per Year
Chief Engineer	1	$ 50,000.
Geologist	1	45,000.
Pit Engineer	1	40,000.
Surveyor	1	32,000.
Surveyor Helper	1	27,000.
Draftsman	1	23,000.
Technician	1	23,000.
TOTAL	7	$ 240,000.
Labour Costs		240,000.
Benefits @ 30%		72,000.
TOTAL		$ 312,000.

COST PER TONNE: $0.0263 (Ore + Waste)

Table 7.6 - Mine Operating Labour Costs

Classification	Number of Personnel	$/Hour	Cost/Year
Shovel Operators	8	$15.00	$ 240,000.
Drill Operators	12	15.00	360,000.
Truck Drivers	32	15.00	960,000.
Auxiliary Equipment Operators	23	15.00	690,000.
Blasting Crew	8	15.00	240,000.
TOTAL	83		$2,490,000.
Labour Cost			2,490,000.
Benefits @ 30%			747,000.
TOTAL			3,237,000.

COST PER TONNE: $0.2725 (Ore + Waste)

Table 7.7 - Mine Maintenance Labour Costs

Classification	Number of Personnel	$/Hour	Cost/Year
Shift Mechanic	12	$15.00	$ 360,000.
Shift Mechanic Relief	3	15.00	90,000.
Shift Electrician	3	15.00	90,000.
Shift Electrician Relief	1	15.00	30,000.
Instrument Man	1	15.00	30,000.
Welder	3	15.00	90,000.
Carpenter	1	15.00	30,000.
Shop Mechanics	8	15.00	240,000.
Helpers	11	11.00	242,000.
TOTAL	43		$1,202,000.
Labour Cost			1,202,000.
Benefits @ 30%			360,600.
			$1,562,600.

COST PER TONNE: $0.1315 (Ore + Waste)

Table 7.8 - Summary of Mine Power Costs

	Total HP	No. of Hours/Year	Cost Per Year
Shovel	1,200	4,685	$ 121,626.
Drill	900	4,685	91,220.
Power Tools	40	1,470	1,272.
Lighting	30 kW	4,200	3,654.
			$ 217,772.

Cost/t Ore + Waste = $0.0183

Table 7.9 - Summary of Direct Milling Costs

Item	Annual Cost	Cost/tonne of Ore
Mill Power	$ 2,413,568.	$ 0.5486
Reagents	1,026,080.	0.2332
Steel	2,716,560.	0.6174
Maintenance Supplies	1,000,000.	0.2273
Heating	1,000,000.	0.2273
Concentrator Labour	3,932,500.	0.8937
TOTAL	$ 12,088,708.	$ 2.7475

Table 7.10 - Summary of Concentrator Power Costs

Item	Total HP	Cost/Year	Cost/Tonne
Primary crushing	1,010	$ 114,714.	$ 0.0261
Fine crushing	1,675	228,292.	0.0519
Grinding	7,490	1,225,012.	0.2784
Flotation/ Dewatering	2,840	464,490.	0.1056
Sub-Total	13,015	2,032,508.	0.4620
Lighting	1500 kW	381,060.	0.0866
TOTAL		2,413,568.	0.5486

Table 7.11 - Concentrator Power Cost Calculations (Rod Mill-Ball Mill Operation)

Process	Cost
Primary Crushing	
(1,010 hp) (0.746) = 753.46 kW	
(753.46 kW) (350 days/yr) (20 hr/day) (0.75 lf) ($0.029/kw)	
$114,714	
$114,714 4,400,000 t/a	$ 0.0261 /t
Fine Crushing	
(1,675 hp) (0.746) = 1,249.55 kW	
(1,249.55 kW) (350) (20) (0.90) lf) ($0.029) = $228,293	
$228,293 4,400,000 t/a	$ 0.0519 /t
Grinding and Classification	
(7490 hp) (0.746) = 5,587.54 kW	
(5,587.54 kW) (350) (24) (0.90 lf) ($0.029) = $1,225,012	
$1,225,012 4,400,000 t/a	$ 0.2784 /t
Flotation/Dewatering	
(2840 hp) (0.746) = 2,118.64 kW	
(2118.64 kW) (350) (24) (0.90 lf) ($0.029) = $464,490	
$464,490 4,400,000 t/a	$ 0.1056 /t

Table 7.12 - Concentrator Flotation Reagent Costs

Reagent	Kg/t Ore	$/Kg Delivered	$/tonne Ore
R343	0.05	2.34	0.1170
MIBC	0.015	1.78	0.0267
S3302	0.005	3.11	0.0156
Lime	0.25	0.19	0.0475
Sodium Cyanide	0.01	2.59	0.0259
Flocculent	0.013*	3.21	0.0005
TOTAL	0.33		0.2332

Annual Cost 4,400,000 t/a x 0.2332 = $1,026,080

Annual Tonnes 4,400,000 t/a x 0.33 = 1,452,300 (1,452 t)

* Consumption = kg/tonne concentrate ≡ 0.00015 kg/t ore

Table 7.13 - Concentrator Steel Consumption and Costs

Item	Kg/t of Ore	Cost/Kg Delivered $	Cost/t of Ore $
Primary crusher	0.004	2.03	0.0081
Cone crushers	0.020	2.17	0.0434
Rod mill liners	0.011	1.23	0.0135
Ball mill liners	0.022	1.20	0.0264
Rods	0.205	0.63	0.1292
Balls	0.515	0.73	0.3760
All other items	0.033	0.63	0.0208
TOTAL	0.810		0.6174

Annual Cost $0.6174 x 4,400,000 = $2,716,560

Annual tonnage $\frac{0.810 \times 4,400,000}{1,000} = 3564$

Table 7.14 - Concentrator Maintenance Supplies

Area	Annual Supplies $	Cost/t of Ore $
Crushing and Stockpile	400,000.	0.0909
Grinding	400,000.	0.0909
Flotation/Dewatering	200,000.	0.0455
TOTAL	$ 1,000,000.	$ 0.2273

Table 7.15 - Summary of Concentrator Supervisory and Labour Costs

Classification	Number of Personnel	Base Cost/Year	Fringe Benefits at 30%	Total Cost/Year	Cost Per Tonne of Ore
Supervision	10	$ 380,000.	$114,000.	$ 494,000.	$ 0.1123
Research and assay office	15	417,000.	125,100.	542,100.	0.1232
Milling labour	48	1,336,000.	400,800.	1,736,800.	0.3947
Maintenance labour	30	892,000.	267,600.	1,159,600.	0.2635
TOTAL DIRECT LABOUR	103	$3,025,000.	$907,500.	$3,932,500.	$ 0.8937

Table 7.16
Milling, Administration, and Supervisory Costs for Concentrator

Classification	Number of Personnel	Unit Rate	Cost Per Year
Mill superintendent	1	$ 50,000.	$ 50,000.
Chief metallurgist	1	45,000.	45,000.
Mill foreman	1	40,000.	40,000.
Shift foreman	4	35,000.	140,000.
Crusher foreman	1	35,000.	35,000.
Repair foreman	1	35,000.	35,000.
Relief foreman	1	35,000.	35,000.
TOTAL	10		$ 380,000.
Salaries per year			
Fringe benefits @ 30%			114,000.
TOTAL PER YEAR			$ 494,000.
Cost Per Tonne of Ore			$ 0.1123

Table 7.17
Costs for Research and Assay Offices for Concentrator

Classification	Number of Personnel	Unit Rate	Cost Per Year
Chief chemist	1	$ 35,000.	$ 35,000.
Chief chemist	4	30,000.	120,000.
Metallurgist	1	40,000.	40,000.
Technician	4	14/hour	112,000.
Sampler	4	11/hour	88,000.
Helper	1	11/hour	22,000.
TOTAL	15		$ 417,000.
Salaries and wages per year			$ 417,000.
Fringe benefits @ 30%			125,100.
TOTAL PER YEAR			$ 542,100.
Cost Per Tonne of Ore			$ 0.1232

Table 7.18
Operating Labour Costs for Concentrator

Classification	Number of Personnel	Hourly Unit Rate	Cost Per Year
Instrument operator	4	$ 15.00	$ 120,000.
Crushing-screening operator	8	15.00	240,000.
Grinding operator	4	15.00	120,000.
Flotation operator	8	15.00	240,000.
Filter operator	4	15.00	120,000.
Tailings operator	4	15.00	120,000.
Loading operator	1	15.00	30,000.
Relief operator	2	15.00	60,000.
Sampler	4	11.00	88,000.
Reagent mixer	1	11.00	22,000.
Helper	8	11.00	176,000.
TOTAL	48		$1,336,000.
Wages per year			$1,336,000.
Fringe benefits at			400,800.
TOTAL PER YEAR			$1,736,800.
Cost Per Tonne of Ore			$ 0.3947

Table 7.19
Maintenance Labour Costs for Concentrator

Classification	Number of Personnel	Hourly Unit Rate	Cost Per Year
Shift mechanic - 1st class	8	16.50	$ 264,000.
Shift mechanic - 2nd class	8	15.00	240,000.
Shift mechanic relief	4	15.00	120,000.
Shift electrican	4	15.00	120,000.
Shift electrican relief	1	15.00	30,000.
Instrument man	1	15.00	30,000.
Helpers	4	11.00	88,000.
TOTAL	30		$ 892,000.
Wages per year			$ 892,000.
Fringe benefits at			267,600.
TOTAL PER YEAR			$1,159,600.
Cost Per Tonne of Ore			$ 0.2635

Table 7.20 - Summary Home Office, Admin., Plant Services Labour

Item	Personnel	Salary	Benefits	Total	$/Tonne
H.O., Admin.	19	680,000.	174,000.	854,000.	0.1941
Plant Services	41	1,119,000.	335,700.	1,454,700.	0.3306
	60	1,799,000.	509,700.	2,308,700.	0.5247

Table 7.21
Home Office and Administration Costs

Classification	Number of Personnel	Unit Rate	Cost Per Year
Head Office			
Manager	1	$ 65,000.	$ 65,000.
Assistant Manager	1	55,000.	55,000.
Chief accountant	1	40,000.	40,000.
Purchasing agent	1	35,000.	35,000.
Payroll accountant	1	35,000.	35,000.
Stores accountant	1	35,000.	35,000.
Personnel director	1	35,000.	35,000.
Stores and shipping clerks	2	15.00	60,000.
Warehousemen	4	12.00	96,000.
Timekeeper	1	12.00	24,000.
Stenos	4	10.00	80,000.
Phone operator	1	10.00	20,000.
TOTAL	19		$ 580,000.
Salaries per year			$ 580,000.
Fringe benefits at 30%			174,000.
Home office per year			100,000.
TOTAL PER YEAR			$ 854,000.
Cost Per Tonne of Ore			$ 0.1941

Table 7.22
Plant Services Cost for Project

Classification	Number of Personnel	Unit Rate	Cost Per Year
Plant engineer	1	$ 45,000.	$ 45,000.
Electrical superintendent	1	40,000.	40,000.
Shop foreman	1	40,000.	40,000.
Electrical foreman	1	40,000.	40,000.
Surface foreman	1	40,000.	40,000.
Safety engineer	1	40,000.	40,000.
Medical officer	1	35,000.	35,000.
First aid attendant	4	12.00	96,000.
Security guard	8	10.00	160,000.
Carpenter	2	15.00	60,000.
Electrician	1	15.00	30,000.
Machinist	2	15.00	60,000.
Pipefitter	1	15.00	30,000.
Mechanic - 1st class	1	16.50	33,000.
Rigger	2	15.00	60,000.
Dozer operator	1	15.00	30,000.
Grader operator	1	15.00	30,000.
Welder	1	15.00	30,000.
Truck driver	4	11.00	88,000.
Maintenance helper	4	11.00	88,000.
Surface labourer	2	11.00	44,000.
TOTAL	41		$1,119,000.
Salaries and wages per year			1,119,000.
Fringe benefits at 30%			335,700.
TOTAL PER YEAR			$1,454,700.
Cost Per Tonne Per Year			$ 0.3306

7.2 Capital Costs

This capital cost estimate is in accordance with the estimate criteria that follow. The design, procurement, expediting and construction would be accomplished by Phillips Barratt Kaiser personnel.

7.2.1 Capital Cost Estimate Summary

7.2.1.1 Column Description

Labour: includes straight time actual craft labour cost up to general foreman.

Burden: includes welfare, insurance, and miscellaneous fringe elements on labour.

Equipment Usage: includes the cost of renting construction equipment and maintaining same.

Material: includes the actual cost of material such as concrete, embedded metal, rebar, tiewire, forms, culverts, grouting, and other materials that become a part of the plant.

Subcontract: consists of those elements that may be subcontracted such as structural steel (fabricated and erected), architectural items such as roofing and siding, heating and ventilating, wallboard, floor tile, acoustics, insulation, electrical installation, mine preproduction stripping.

Equipment: represents equipment purchased and installed.

7.2.1.2 Line Numbers and Description Columns (Direct Costs)

Direct costs include equipment purchase, freight, British Columbia tax, materials costs, equipment rental for construction, labour and burden to install.

Line 7, General Site and Utilities: includes soils investigation, site surveys, cut and fill, culverts, underground disposal lines, miscellaneous structures and plant storage tanks.

Line 10, Crushing and Stockpile: includes structures and numbered equipment in the crushing and stockpile area up to, but not including the fine ore bins.

Line 11, Grinding: includes numbered equipment in the grinding.

Line 12, Flotation and Dewatering Area; includes the balance of the process plant equipment.

Line 14, Mine, Plant Service Administration Building: includes repair shops, machine shops and warehouse, offices for mine and mill, change rooms and equipment including cranes.

Line 16, Mobile Equipment: is as selected.

Line 17, Piping: includes all distribution lines for air, water and steam, and all process lines carrying product between flotation heads and tails, within the plant boundary.

Line 18, Electrical: includes power supply, plant distribution to mine and mill and to reclaim, at line voltage per single line diagram. Motor control centers and switchgear are a part of this item as are all conduit and wire.

Line 19, Instrumentation: is an allowance based on current industry practice.

Line 20, Preoperational Testing: includes that effort required to operate equipment on a dry-run basis, to effect calibration as required, and to test interlocks and panels.

Line 24, Taxes: include fixed taxes on materials and subcontract elements required on construction.

Line 25, Escalation: is calculated from the direct cost of the crafts concerned with the job and their existing contracts. The escalation is carried on through construction. Escalation is also based on the construction schedule and its extended program for high peaks in summer and low peaks in winter. There is also an escalation of 5% on equipment and 10% on contractors' field overhead. Engineering, supervision and procurement are escalated at 10%.

7.2.1.3 Line Numbers and Description Columns (Indirect Costs)

Indirect costs include items which support the direct costs but are not a part of the physical plant.

Line 31, Contractors' Field Overhead and Profit: includes salaries and burden for supervisors, engineers, accounting, purchasing and other costs required to construct the plant. Move-in expenses and travel expenses are included for field staff personnel.

Line 32, Construction Plant: includes the tools, warehouse, shops, and consumable supplies required for construction including utilities and power.

Line 36, Engineering, Procurement and Construction Management (EPCM): covers costs of producing drawings, specifications, purchase and expediting of equipment, project costs, communications and travel for engineering personnel.

7.2.2 Estimate Criteria

1. The estimate is based on the following schedule:

   Engineering will commence during March, 1989 and will be substantially complete by June, 1990.

   Construction will start approximately June 1, 1989, and will be ready for startup by July 15, 1991.

2. Design is based on a standard one shift work week. Construction is based on a one shift 60-hour week.

3. The estimate is based on labour and material prices as of March, 1988.

4. Escalation has been included as a line item based on the schedule shown above.

5. No allowances are included for possible increased costs due to shortage of skilled craft labour or possible delay in material and/or equipment deliveries.

6. Startup and training of operating personnel are not included.

7. The following items are not included:

   a. Owners cost for engineering, general services and expenses.

   b. Communications - other than in-plant.

   c. Owner furnished soil analysis, survey and test work.

   d. Allowance for work stoppage due to union conflicts.

7.2.3 Summary of Engineering Work by PBK

The following is a summary of the work performed by PBK in relation to this prefeasibility engineering study and the capital and operating cost estimates for direct cost items:

1. Mine preproduction stripping.

2. Plant surface drainage and grading including plant roads and parking areas and including culverts as required.

3. All buildings in the plant area including heating, ventilating, plumbing and lighting.

4. All process plant equipment including piping, electrical, instrumentation and controls in accordance with the published flowsheets and the equipment lists.

5. Main power line, water supply, fuel storage, boiler section for plant heat and distribution lines to all plant load centres, the mine, the tailings dam.

6. Water storage and distribution at the process plant including fire water system.

7. Tailings disposal line to tailings pond.

8. Sanitary sewage system for the plant including the lagoons.

9. Construction facilities, including grading and drainage, sanitary sewage system, water, fire protection and water storage and electrical.

10. Gasoline and diesel oil dispensing system at the plant for mobile operating equipment.

11. All shop equipment.

12. Equipment for both metallurgical and analytical laboratories.

13. Mine, plant mobile equipment.

The following estimate is for conventional grinding and an open air coarse ore stockpile.

Phillips Barratt Kaiser Engineering Ltd.

PROJECT TITLE: HAIL HARPER CREEK - BASE CASE
CLIENT: AURUN MINES LTD
LOCATION: VAVENBY, BC

JOB NO. 88005
DATE 88/03/29 BY GGW
SHEET 1 OF 1
CHECKED BY

	ACCT	REF	DESCRIPTION	QUANTITY	UNIT MH	UNIT EU	UNIT M	UNIT SC	MANHOUR	LABOR RATES	LABOR & BURDEN	EQUIPMENT USAGE	MATERIAL	SUB-CONTRACT	ALLOWANCE	EQUIPMENT	TOTAL
1			CAPITAL COSTS														
2			DIRECT COSTS														
3	0100	0000	Access Road Upgrading											1 729 000			1 729 000
4	0200	0000	Mine Waste Dump / Tailings Starter Dam											1 973 000			1 973 000
5	0300	0000	Mine Preproduction Stripping											1 400 000			1 400 000
6	0400	0000	Site Clearing & Grading											875 000			875 000
7	0500	0000	Site Services / Utilities								70 000	10 000	50 000	2 315 000		148 000	2 593 000
8	0600	0000	Water Supply								70 000	10 000	100 000	2 290 000		530 000	3 000 000
9	0700	0000	Mine Mobile Equipment.								—	—	—	—		16 780 000	16 780 000
10	0800	0000	Crushing & Stockpile Area								2 444 000	260 000	1 945 000	5 387 000		4 898 000	14 924 000
11	0900	0000	Grinding Area								1 542 000	200 000	1 129 000	4 201 000		4 995 000	12 067 000
12	1000	0000	Flotation / Dewatering Area								286 000	75 000	—	681 000		2 024 000	3 066 000
13	1100	0000	Tailings / Reclaim Pumping / Piping								70 000	10 000	210 000	1 309 000		496 000	2 095 000
14	1200	0000	Mine / Plant Services, Admin Buildings								681 000	65 000	738 000	3 392 000		1 024 000	5 900 000
15	1300	0000	Plant Heating												750 000		750 000
16	1400	0000	Plant Mobile Equipment													600 000	600 000
17	1500	0000	Piping												6 020 000		6 020 000
18	1600	0000	Electrical												9 902 000		9 902 000
19	1700	0000	Instrumentation												1 192 000		1 192 000
20	1800	0000	Preoperational Testing												250 000		250 000
21	1900	0000	Mine / Mill Initial Spares												1 434 000		1 434 000
23			Sub Total								5 163 000	620 000	4 172 000	25 552 000	19 548 000	31 495 000	86 550 000
24	6000	0000	Taxes (FST 12% PST 6%)										781 000	675 000	1 450 000	1 889 000	4 795 000
25	6100	0000	ESCALATION												12 550 000		10 070 000
27			TOTAL DIRECT COSTS.								5 163 000	620 000	4 953 000	26 227 000	33 550 000	33 384 000	101 415 000
29			INDIRECT COSTS														
31	7000	0000	Contractors Overheads & Profit														8 736 000
32	7100	0000	Construction Plant														600 000
34			TOTAL INDIRECTS														9 336 000
35			TOTAL CONSTRUCTION COSTS.														110 771 000
36	8000	0000	E.P.C.M. (Including Fees)														10 140 000
38			Sub Total														120 911 000
39	9000	0000	CONTINGENCY														18 136 000
40			PROJECT TOTAL CAPITAL COSTS														139 047 000

Phillips Barratt Kaiser
Engineering Ltd

EQUIPMENT LIST

Project No: 88005
Date: 15 March/88
Revision: 0

Project: AURUN HAIL HARPER CREEK

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	HORSEPOWER NO	HORSEPOWER EA	HORSEPOWER TOTAL	REMARKS	$ CAPITAL COST
CR-01	Primary Crusher	54"	1600t/h	--	1	500	500	Gyratory	1 100 000
BN-01	Surge Hopper	–	250t		–	–	–		–
FE-01	Pan Feeder	60"	1600t/h		1	60	60		65 000
CV-01	Conveyor #1	54"	1600t/h		1	150	150		195 000
SC-01	Primary Screen.	8'x20'	1600t/h		1	40	40	Double Deck	49 000
CV-02	Conveyor #2	30"	210t/h		1	25	25		14 000
CV-03	Conveyor #3	48"	1400t/h.		1	125	125		250 000
FE 02-05	Reclaim Feeders	36"x60"			4	15	60	Vibrating type	36 000
CV-04	Conveyor #4	42"	1100t/h		1	75	75	1010	191 000
CR-02	Secondary Crusher	7'	900t/h		1	350	350	Standard	550 000
CV-05	Conveyor #5	48"	1950t/h		1	15	15		22 000
CV-06	Conveyor #6	48"	1950t/h.		1	150	150		170 000
CV-07	Conveyor #7	48"	1950t/h.		1	150	150		170 000
BN-02	Tertiary Feed Bin	—	1950t/h.		—	—	—	Flow splitter	50 000

Phillips Barratt Kaiser
Engineering Ltd

EQUIPMENT LIST

Project No: 88005
Date: 15 March/88
Revision: 0

Project: AURUN HAIL HARPER CREEK

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	HORSEPOWER NO	HORSEPOWER EA	HORSEPOWER TOTAL	REMARKS	$ CAPITAL COST
FE 06-07	Tertiary Feeders	60"x96"	1100t/h.		2	30	60		28 000
SC 02-03	Secondary Screen	8'x20'	600 t/h.		2	40	80	double deck	98 000
CR-0304	Tertiary Crushers	7'			2	350	700	Shorthead	1 200 000
CV08	Conveyor #8	42"	1300t/h.		1	125	125		270 000
CV09	Conveyor #9	48"	1300t/h.		1	20	20		90 000
TR-01	Fine Ore Tripper	48"	1300t/h		1	25	25		INCL.
BN-0306	Fine Ore Bins	—	2500t		4	—	—	heated	750 000
FE 08-15	Slot Feeders	12'x10"-26"	150t/h.		8	25	200		160 000
CV-10	Conveyor #10	48"	600t/h		1	15	15		22 000
CV-11	Conveyor #11	48"	600t/h		1	15	15		22 000
SP-01	Primary Crush. Sump	3"			1	25	25	Vertical pump	5 000
SP-02	Secondary Crush Sump	3"			1	25	25	Vertical pump	5 000
SP-03	Fine Ore Sump	3"			1	25	25	Vertical pump	5 000
DC-01	Primary Dust System	—	100,000 m^3/h.		1	75	75		150 000

Phillips Barratt Kaiser
Engineering Ltd

EQUIPMENT LIST

Project: AURUN HAIL HARPER CREEK

Project No: 88005
Date: 15 March/88
Revision: 0

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	HORSEPOWER NO	EA	TOTAL	REMARKS	$ CAPITAL COST
DC-02	Secondary Dust Sys		176 000 m³/h.		1	150	150		200 000
ML-01	Rod Mill	13'ø x 21'	550 t/h		1	1600	1600		1 050 000
ML-0203	Ball Mills	16'ø x 16'	275 t/h.		2	2500	5000		2 760 000
PB-0102	Ball Mill Pumpbox	20 m³			–	–	–		15 000
PP-01-06	Ball Mill Disch Pumps	12"x14"			6	60	360		87 000
PB-03	FLOTATION Feed P.B				–	–	–		15 000
PP-0708	Flotation Fd Pumps	12x14			2	75	150		29 000
CY01-10	Ball Mill Cyclones	30'			–	–	–		70 000
TK-0102	Rougher Conditioner	10'ø x 12'			2	–	–		10 000
Ag0102	Rough. Cond. Agitator				2	40	80		52 000
FC01-24	Rougher Flotation Cells		500 ft³		24	40	960		975 000
PB-04	Tailings Pumpbox	20 m³			–	–	–		15 000
PP-09-11	Tailings Pumps	12"x14"			3	200	600		43 000
[illegible]	[illegible]	[illegible]				–	–		3 000

Phillips Barratt Kaiser
Engineering Ltd

EQUIPMENT LIST

Project: AURUN HAIL HARPER CREEK

Project No: 88005
Date: 15 March/88
Revision: 0

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	NO	HORSEPOWER EA	HORSEPOWER TOTAL	REMARKS	CAPITAL $ COST
PP12-13	Rougher Con. Pumps	4x3			2	15	30		7 000
PB06	Regrind Pumpbox	1.5m³			1	—	—		3 000
PP14-15	Regrind Pumps	4x3			2	10	20		7 000
PB07	First Cl. Feed Pumpbox	1.5m³			1	—	—		3 000
PP16,17	First Cl Feed Pumps	4x3			2	15	30		7000
CY11-14	Regrind Cyclones	12"ø			4	—	—		15 000
TK-02	1st Cleaner Conditioner	8'ø x 9'	[illegible]						20 000
CF25-30	1st Cl. Flotation Cells		100 ft³		3	20	60	1 drive/2 cells	75 000
CF31-36	1st Cl Scav. Flot Cells		100 ft³		[illegible]	[illegible]	[illegible]	"	75 000
PB-08	1st Cl Conc Pumpbox	[illegible]			—	—	—		2 000
PP[illegible]	1st Cl [illegible]	[illegible]			2	3	6		6 000
CF37-41	2nd Cleaner Float Cells		40 ft³		3	15	45	1 drive/2 cells	70 000
PB-09	2nd Cleaner Tails Pumpbox	[illegible]			—	—	—		2 000
PP[illegible]	2nd Cl Tails [illegible]	[illegible]			2	3	6		6 000

Phillips Barrett Kaiser
Engineering Ltd

EQUIPMENT LIST

Project: AURUN HAIL HARPER CREEK

Project No: 88005
Date: 15 March/88
Revision: 0

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	HORSEPOWER NO	HORSEPOWER EA	HORSEPOWER TOTAL	REMARKS	$ CAPITAL COST
DC-03	Lime Dust Collector		1000 cfm						10 000
ML-04	Lime Ball Mill				15	1	15		12 000
TK-06	Lime Storage Tk.	10'Øx10'			—	—	—		7 000
AG-06	Agitator.				1	10	10		14 000
PP 30-31	Lime Circulating Pumps	2x2		SRL	2	5	10		6 000
RE 01	R343 Reagent Sys.	—			2	0.5	1.0		12 000
RE 02	[illegible] Sys	—			2	0.5	1.0		12 000
RE 03	[illegible] Sys	—			2	0.5	1.0		12 000
SC 01	TRUCK SCALE		100 t						35 000

Phillips Barratt Kaiser
Engineering Ltd

EQUIPMENT LIST

Project: AURUN HAIL HARPER CREEK

Project No: 88005
Date: 15 March/88
Revision: 0

ITEM NUMBER	DESCRIPTION	SIZE	CAPACITY	MAT'LS of CONSTR	HORSEPOWER NO	HORSEPOWER EA	HORSEPOWER TOTAL	REMARKS	$ CAPITAL COST
PB10	[illegible] Conc Pumpbox	$[illegible] m^3$			–	–	–		2 000
PF2[illegible]	[illegible] Conc Pumps	2x2			2	3	6		6 000
TH-01	Concentrate Thickener	16ft.d			2	3	6	High Capacity	130 000
PP24-[illegible]	Thickener O/Flow Pumps	2x2			2	3	6		6 000
TK-04	Thick U/Flow Surge Tk	10'dx10'			–	–	–		10 000
AG-[illegible]	Agitator.				1	15	15		16 000
PF26-27	Conc Filter Feed Pumps	2x2.			2	3	6		6 000
FL01	Conc Filter	300ft²			2	1.5	3	Disc	75 000
VP01	Vacuum [illegible]				1	25	25		30 000
[illegible]	Filtrate Pumps	2x2			2	1.5	3		6 000
[illegible]	[illegible] Blower		10,000 cfm		2	125	250		67 000
TK-05	Process Water Stg Tk	42'd x [illegible]	[illegible]		1	–	–	–	120 000
PW-01/02[illegible]	Process Water Pumps				3	125	375		17 000
BN-07	Lime Storage Bin	–	30t		–	–	–		17 000

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005
AURUN MINES LTD.
HAIL HARPER CREEK

Estimated by: BC + LP **Date:** Mar 30, 1988
Ref. Dwgs: 88005 - 6.1 + 6.2
SUMMARY - CIVIL

Item Number	Description	Qty	Unit	Unit Cost	Total
1.	Lined ditch at dam				37,000
2.	Lined ditch at waste rock dump				112,000
3.	Unlined ditches				158,000
4.	Piping acid wastewater to treatment				216,000
5.	Settling ponds				245,000
6.	Treatment basin				154,000
7	Settling pond outlet ditches				18,000
8.	Culverts				33,000
9	Access road upgrading				1,729,000
10.	Site clearing and grading				875,000
11.	Fire protection				78,000
12.	Tanks				536,000
13.	Tailings Disposal and Reclaim Lines				970,000
14.	Pumping at Tailings Pond				1,125,000
15.	Potable Water System				65,000
16.	Sewage System				123,000
17.	Employee and visitor parking				43,000
18.	Access roads				945,000
19.	Water supply				3,000,000
20.	Diesel Tank Farm				803,000
					11,265,000

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005
HAIL HARPER CREEK
AURUN MINES LTD.

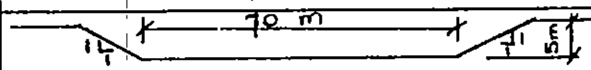
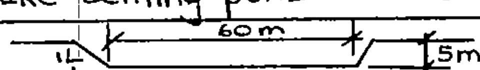
Estimated by: BC. + L.P. **Date:** Mar. 30, 1988
Ref. Dwgs: 88005-6.1 + 6.2

Item Number	Description	Qty	Unit	Unit Cost	Total
1	Lined ditch to collect dam seepage	600	m	62.00	37,000
	- section 1m 1 $A = 4\ m^3/m$				
	$4\ m^3/m \times \$3/m^3 = \$12/m$ for ditch				
	- liner cost \$50/m = \$62/m total				
2.	Lined ditch to collect waste rock seepage	1800	m	62.00	112,000
3.	Unlined ditches to collect surface runoff	13,200	m	12.00	158,000
	around: waste rock dump area; west and				
	east ore zones, plant site. (To settling ponds)				
4.	Piping of water from dam and waste	2000	m	108.00	216,000
	rock dump area to treatment basin.				
	Area $= \frac{1}{2}(1400 \times 1400) = 1 \times 10^6\ m^2 = 100\ ha$				
	Assume $I = 1''/hr$ (25 mm/hr) $C = 0.25$				
	$Q = \frac{1000}{360} CIA = \frac{1000}{360}(0.25)(25)(100)$				
	$Q = 1750\ \ell/s$; $S \approx 8\%$ $\therefore D = 600\ mm$				
5.	Settling ponds	28,000	m^3	5.00	140,000
a)	From waste rock dump area				
	- Assume max. storage for 6 hr. detention.				
	$Q = \frac{1000}{360} CIA$ $c = 0.25$ $A = 5 \times 10^6\ m^2 = 500\ ha$				
	Assume a peak daily flow of 80 mm (10 yr.				
	return period) → $I = 3.5\ mm/hr$				
	$Q = \frac{1000}{360}(0.25)(3.5)(500) = 1200\ \ell/s$				
	$1200\ \ell/s \times \frac{3600\ s}{hr} \times 6\ hr = 25 \times 10^6\ \ell = 25000\ m^3$				

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005 **Estimated by:** **Date:**
Ref. Dwgs:

Item Number	Description	Qty	Unit	Unit Cost	Total
--	Make settling pond 5 m × 70 m × 70 m				
					
	Vol. of excavation = (70×70×5) +				
	$[\frac{1}{2}(5\times5)\times(70\times4)] = 28{,}000\ m^3$				
5. b)	From plant site and ore zones	21,000	m^3	5.00	105,000
	$A = 1500\times400 + \frac{1}{2}(1100\times400)$				
	$+\frac{1}{2}(300\times400) + 1800\times500$				
	$+\frac{1}{2}(1500\times600) + \frac{1}{2}(500\times400)$				
	$+1000\times300 + \frac{1}{2}(400\times300)$				
	$+600\times300 = 3\times10^6\ m^2 = 300\ ha$				
	$Q = \frac{1000}{360}(0.25)(3.5)(300) = 730\ \ell/s$				
	$730\ \ell/s \times \frac{3600s}{hr} \times 6hr = 16{,}000\ m^3$				
	Make settling pond 5 m×60 m×60 m				
					
	Vol. of excavation = (60×60×5)				
	$+[\frac{1}{2}(5\times5)\times(60\times4)] = 21{,}000\ m^3$				
6.	Basin for treatment of acidic runoff				
	Assume concrete - 250 mm walls				
	Assume storage for 12 hr. - i.e. water				
	is treated and released twice daily.				
	Assume Q from dam is 10% that				
	of waste rock dump area (cutoff in dam).				

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005 **Estimated by:** **Date:**
Ref. Dwgs:

Item Number	Description	Qty	Unit	Unit Cost	Total
--	Assume C for waste rock dump = 0.35				
	$Q = \frac{1000}{360}(0.35)(3.5)(\underset{Area}{100}) = 340\ \ell/s$				
	$340\ \ell/s \times 3600\ sec/hr \times 12\ hr = 15000\ m^3$				
	Make detention basin 5m × 55m × 55m				
	Concrete volume = (55 × 5 × .25) × 2				
	+ 4.5 × .25 × 2 = 140 m^3	140	m^3	350.00	49,000
	Excavation vol. = (55.5 × 55.5) × 2 +				
	$[\frac{1}{2}(5.25)^2 \times (55.5 \times 4)]$ = 19200 m^3	19200	m^3	5.00	96,000
	Tank volume (outside) = 55.5 × 55.5 × 5.25				
	= 16,200 m^3				
	∴ Backfill volume = 3000 m^3	3000	m^3	3.00	9,000
7.	Ditches from settling ponds to creeks	1500	m	12.00	18,000
8.	Culverts	210	m	150.00	32,000
	Headwalls	10	ea	100.00	1000
9.	Access road upgrading				
	- assume 10 m width				
	14 km @ 10 m wide = 140,000 m^2				
	- assume 5 km will require clearing				
	and grubbing as well as grading	5000	m	110.00	550,000
	- base course gravel for 14 km	14000	m	25.25	354,000
	Assume 11 km requires grading only.	11,000	m	75.00	825,000
10. a)	Site grading	35	ha	22,000	770,000
b)	Site clearing	35	ha	3,000	105,000

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005 **Estimated by:** **Date:**
Ref. Dwgs:

Item Number	Description	Qty	Unit	Unit Cost	Total
11.	Fire Protection System				
a)	Fire main - 8"	775	m	75.00	58,000
b)	Hydrants	10	ea	1300.00	13,000
c)	PIV's	6	ea	1200.00	7,000
12.	Tanks				
a)	Process water	2,300,000	ℓ	0.12	276,000
b)	Fire water	1,400,000	ℓ	0.14	196,000
c)	Potable water	230,000	ℓ	0.28	64,000
13.	Tailings Disposal and Reclaim Lines				
a)	Disposal - 4810 USgpm → 18" ϕ } use				
	Return - 3000 USgpm → 16" ϕ } 18" ϕ for both				
	Lines from plant to dam:				
	3-18" ϕ Sclairpipe lines (spare disposal line) with weights	2400	m	350.00	840,000
b)	Disposal line in pond				
	18" ϕ Sclairpipe with weights	1000	m	130.00	130,000
14.	Pumping at Tailings Pond				
	3 pumps - 2 duty and 1 spare, ea. 750hp	2250	hp	500.00	1,125,000
15.	Potable Water System				
	4" ϕ D.I. line, 2.4 m cover	750	m	86.00	65,000
16.	Sewage System				
a)	sanitary sewers - 200 ϕ	820	m	50.00	41,000
b)	manholes	7	ea	1000.00	7,000

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.
Phillips Barratt Engineers and Architects

Project Number: 88005 Estimated by: Date:
Ref. Dwgs:

Item Number	Description	Qty	Unit	Unit Cost	Total
c)	Treatment plant - extended aeration	LS			66,000
d)	Tile disposal field	LS			15,000
17.	Employee and visitor parking				
	assume 100 mm of pit run	9600	m^2	2.00	19,000
	and 100 mm of base course	9600	m^2	2.50	24,000
18.	Access roads to explosives building,				
	tailings dam and waste rock dump.				
	Assume 10 m width				
a)	Clearing + grubbing	7000	m	35.00	245,000
b)	Grading	7000	m	75.00	525,000
c)	Base course	7000	m	25.00	175,000
19.	Water Supply				
a)	River intake	LS			30,000
b)	Settling basin	LS			120,000
c)	River pumping station, 3x500 hp pumps	LS			750,000
d)	Booster pumping station, 3x500 hp pumps	LS			750,000
e)	Pipeline - 10" ø steel, 8.25 km	LS			1,350,000
20.	Diesel Tank Farm				
a)	Tanks	2	ea	380,000	760,000
b)	Piping and dispensers	LS			20,000
c)	Dykes and liner	LS			23,000

COST ESTIMATE

Phillips Barratt Kaiser Engineering Ltd.

Phillips Barratt Engineers and Architects

Project Number: 88005 — Estimated by: M. SHERIFF — Date: 88-03-30

AURUN MINES LTD. — Ref. Dwgs:

HAIL HARPER CREEK

Item Number	Description	Qty	Unit	Unit Cost	Total
1.	BC HYDRO FEE				1 000 000
2.	138 kV SWITCHGEAR & EQUIPMENT				300 000
3.	15/20 MVA TRANSFORMER				375 000
4.	13.8 kV SWITCHGEAR				225 000
5.	STATION SERVICE & AUXILIARY				30 000
6.	MILLING & CRUSHING				4 672 000
7.	OPEN PIT 4000 kVA				2 400 000
8.	TAILINGS & RECLAIM				800 000
9.	ADMINISTRATION & LABORATORYS				100 000
	TOTAL ELECTRICAL				9 902 000
	LESS BC HYDRO FEE				1 000 000
					8 902 000
	CONTINGENCY FOR LAB & MAT. 15%				1 335 000
					10 237 000
	PLUS BC HYDRO FEE				1 000 000
					11 237 000

Future Drilling Programme
Cost Estimate

1.0 Grand Total

Wages	1 geologist drill supervision core logging sample preparation	150 days	@ $ 250 /day	$ 37,500.
Food & Accommodation		150 days	@ $ 60 /day	9,000.
Transportation	4 x 4 vehicles	150 days	@ $ 50 /day	7,500.
Drilling	mob-demob.			50,000.
	drilling E Zone (NQ)	9,100 m	@ $ 45 /m	409,500.
	drilling W Zone	5,200 m	@ $ 45 /m	234,000.
Core Boxes		2,000 boxes	@ $7.50 /box	15,000.
Analyses	3m samples copper assays	1,200 assays	@ $ 10 /sample	12,000.
Miscellaneous	(supplies (freight (long distance calls			2,000.
Report Writing		15 days	@ $ 250	3,750.
Head Office Expenses				5,000.
				$785,250.
			Say	**$800,000.**

2.0 Requirements for Definitive Feasibility Study

Drilling: 6090 m $\frac{6090}{14,300} \times \$800,000$ **$ 341,000**

3.0 Requirements for Exploration

$800,000 - 341,000 **$ 459,000**

Note:
Above estimate assumes use of Aurun Mines Ltd.'s permanent personnel in technical and supervisory functions.

DRILLING PROGRAMME W. ZONE (all holes are vertical)

Main Sections	Grid filling	Drilling extension down to level 1500 m	North extension of the grid	Exploration to the North	
600 W 150 m					
750 W 120 m					
870 W 120 m		700 N - 240 m			
990 W 60 m				800 N - 340 m	
1050 W 60 m	280 N - 50 m	450 N - 170 m 550 N - 230 m	650 N - 280 m 750 N - 320 m	1000 N - 380 m (1200 N - 430 m)	
1110 W 60 m			650 N - 260 m	800 N - 310 m	
1170 W 90 m	300 N - 120 m	450 N - 170 m 550 N - 250 m			
1260 W 60 m		deepening NH-54 90 m NH-53 120 m	550 N - 280 m		
1320 W 60 m		100 N - 190 m deepening NH-74 150 m 500 N - 250 m			
1380 W 60 m		550 N - 250 m			
1440 W	150 N - 50 m	500 N - 240 m			
TOTAL	220 m	2350 m	1140 m	1460 m	5170 m

DRILLING PROGRAMME E. ZONE (all holes are vertical except if indicated otherwise)

Main Sections	Grid filling	Drilling extension down to level 1500 m	North extension of the grid	Exploration: Exploration to the North	Exploration: Exploration between Zone N. & S	
420 E 90 m	50 N - 220 m		300 N - 300 m 500 N - 500 m			
330 E 90 m		300 N - 250 m 390 N - 250 m	480 N - 250 m 700 N- 310 m	360 E 1000 N - 370 m (1200 N - 430 m)		
240 E 90 m	350 N - 400 m		800 N - 350 m			
150 E 30 m		500 N - 250 m				
60 E 60 m			800 N - 480 m			
OBL 60 m	100S - 50 m BL - 130 m 100N - 200 m 200N - 200 m	300 N - 200 m 400 N - 230 m	500 N - 260 m 600 N - 300 m 800 N - 370 m			
60 W 60 m	200 N (60°) - 200 m		600 N - 290 m		3505 - 150 m	
120 W 60 m	BL - 150 m 100 N - 170 m 200 N - 200 m					
180 W 60 m		400 N - 200 m	600 N - 270 m			
240 W 60 m			400 N - 280 m			
300 W	BL - 100 m 100N - 170 m	200 N - 200 m	300 N - 200 m		3505 - 200 m	
TOTALS	**2190 m**	**1580 m**	**4160 m**	**800 m**	**350 m**	**9080 m**

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(102,745,410)		(500,000)									SALVAGE 3,469,596
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			76,050,252	76,050,252	64,642,714	64,642,714	64,642,714	64,642,714	64,642,714	64,642,714	76,050,252	76,050,252	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			80,116,173	80,116,173	68,098,747	68,098,747	68,098,747	68,098,747	68,098,747	68,098,747	80,116,173	80,116,173	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			36,848,493	25,793,945	18,055,762	12,639,033	8,847,323	6,193,126	4,335,188	3,034,632	2,124,242	1,486,970	
royalty			2,403,485	2,403,485	2,042,962	2,042,962	2,042,962	2,042,962	2,042,962	2,042,962	2,403,485	2,403,485	
TOTAL OPERATING COSTS			64,146,667	53,492,988	46,853,551	40,192,523	38,957,510	36,848,786	34,990,848	34,235,765	44,317,393	46,057,552	
OPERATING PROFIT			15,969,506	26,623,185	21,245,196	27,906,225	29,141,237	31,249,961	33,107,899	33,862,983	35,798,780	34,058,622	
TAXES AT 57%			9,102,619	15,175,216	12,109,762	15,906,548	16,610,505	17,812,478	18,871,503	19,301,900	20,405,305	19,413,414	
NET INCOME			6,866,888	11,447,970	9,135,434	11,999,677	12,530,732	13,437,483	14,236,397	14,561,083	15,393,475	14,645,207	
CASH FLOW	(20,082,900)	(102,745,410)	43,715,381	36,741,915	27,191,196	24,638,710	21,378,055	19,630,610	18,571,585	17,595,714	17,517,718	16,132,177	3,469,596
CUMULATIVE CASH FLOW	(20,082,900)	(122,828,310)	(79,112,929)	(42,371,015)	(15,179,819)	9,458,891	30,836,946	50,467,556	69,039,141	86,634,855	104,152,573	120,284,750	123,754,346
DEPRECIATED VALUE			85,979,817	60,185,872	42,130,110	29,491,077	20,643,754	14,450,628	10,115,439	7,080,808	4,956,565	3,469,596	

DCF-ROI	18.25
NPV10%	32,633,295
NPV12.5%	20,197,794
NPV15%	10,195,142
PAYBACK (YEARS)	3.62
Cu,$US/lb.	2.00
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

USED EQUIPMENT; $US 2.00/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(102,745,410)		(500,000)									SALVAGE 3,469,596
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			41,067,136	41,067,136	34,907,066	34,907,066	34,907,066	34,907,066	34,907,066	34,907,066	41,067,136	41,067,136	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			45,133,057	45,133,057	38,363,099	38,363,099	38,363,099	38,363,099	38,363,099	38,363,099	45,133,057	45,133,057	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			36,848,493	25,793,945	18,055,762	12,639,033	8,847,323	6,193,126	4,335,188	3,034,632	2,124,242	1,486,970	
royalty			1,353,992	1,353,992	1,150,893	1,150,893	1,150,893	1,150,893	1,150,893	1,150,893	1,353,992	1,353,992	
TOTAL OPERATING COSTS			63,097,174	52,443,495	45,961,482	39,300,453	38,065,441	35,956,717	34,098,779	33,343,695	43,267,900	45,008,058	
OPERATING PROFIT			(17,964,116)	(7,310,437)	(7,598,383)	(937,354)	297,658	2,406,382	4,264,320	5,019,404	1,865,157	124,999	
TAXES AT 57%			(10,239,546)	(4,166,949)	(4,331,078)	(534,292)	169,665	1,371,638	2,430,662	2,861,060	1,063,140	71,250	
NET INCOME			(7,724,570)	(3,143,488)	(3,267,305)	(403,062)	127,993	1,034,744	1,833,658	2,158,344	802,018	53,750	
CASH FLOW	(20,082,900)	(102,745,410)	29,123,923	22,150,457	14,788,457	12,235,971	8,975,316	7,227,871	6,168,846	5,192,975	2,926,260	1,540,719	3,469,596
CUMULATIVE CASH FLOW	(20,082,900)	(122,828,310)	(93,704,387)	(71,553,930)	(56,765,473)	(44,529,502)	(35,554,186)	(28,326,315)	(22,157,469)	(16,964,494)	(14,038,234)	(12,497,515)	(9,027,919)
DEPRECIATED VALUE			85,979,817	60,185,872	42,130,110	29,491,077	20,643,754	14,450,628	10,115,439	7,080,808	4,956,565	3,469,596	

DCF-ROI	-1.93
NPV10%	###########
NPV12.5%	###########
NPV15%	###########
PAYBACK (YEARS)	0.00
Cu,$US/lb.	1.08
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

USED EQUIPMENT; $US 1.08/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	5.65	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			76,050,252	76,050,252	64,642,714	64,642,714	64,642,714	64,642,714	64,642,714	64,642,714	76,050,252	76,050,252	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			80,116,173	80,116,173	68,098,747	68,098,747	68,098,747	68,098,747	68,098,747	68,098,747	80,116,173	80,116,173	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			2,403,485	2,403,485	2,042,962	2,042,962	2,042,962	2,042,962	2,042,962	2,042,962	2,403,485	2,403,485	
TOTAL OPERATING COSTS			67,313,974	55,710,103	48,405,531	41,278,909	39,717,981	37,381,115	35,363,479	34,496,606	44,499,982	46,185,364	
OPERATING PROFIT			12,802,199	24,406,070	19,693,216	26,819,839	28,380,767	30,717,632	32,735,269	33,602,141	35,616,191	33,930,809	
TAXES AT 57%			7,297,254	13,911,460	11,225,133	15,287,308	16,177,037	17,509,050	18,659,103	19,153,221	20,301,229	19,340,561	
NET INCOME			5,504,946	10,494,610	8,468,083	11,532,531	12,203,730	13,208,582	14,076,166	14,448,921	15,314,962	14,590,248	
CASH FLOW	(20,082,900)	(113,303,100)	45,520,746	38,005,670	28,075,825	25,257,950	21,811,523	19,934,037	18,783,984	17,744,394	17,621,793	16,205,030	3,767,824
CUMULATIVE CASH FLOW	(20,082,900)	(133,386,000)	(87,865,254)	(49,859,584)	(21,783,759)	3,474,191	25,285,714	45,219,751	64,003,736	81,748,130	99,369,923	115,574,953	119,342,777
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	16.58
NPV10%	27,683,788
NPV12.5%	15,247,600
NPV15%	5,272,890
PAYBACK (YEARS)	3.86
Cu,$US/lb.	2.00
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 2.00/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%) copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			5.09	5.09	5.09	5.09	5.09	5.09	5.09	5.09	5.09	5.09	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			68,445,227	68,445,227	58,178,443	58,178,443	58,178,443	58,178,443	58,178,443	58,178,443	68,445,227	68,445,227	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,983	
TOTAL SALES REVENUE			72,511,148	72,511,148	61,634,476	61,634,476	61,634,476	61,634,476	61,634,476	61,634,476	72,511,148	72,511,148	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			2,175,334	2,175,334	1,849,034	1,849,034	1,849,034	1,849,034	1,849,034	1,849,034	2,175,334	2,175,334	
TOTAL OPERATING COSTS			67,085,823	55,481,952	48,211,603	41,084,981	39,524,052	37,187,187	35,169,551	34,302,678	44,271,832	45,957,213	
OPERATING PROFIT			5,425,325	17,029,196	13,422,873	20,549,495	22,110,424	24,447,289	26,464,925	27,331,798	28,239,317	26,553,935	
TAXES AT 57%			3,092,435	9,706,642	7,651,037	11,713,212	12,602,941	13,934,955	15,085,007	15,579,125	16,096,410	15,135,743	
NET INCOME			2,332,890	7,322,554	5,771,835	8,836,283	9,507,482	10,512,334	11,379,918	11,752,673	12,142,906	11,418,192	
CASH FLOW	(20,082,900)	(113,303,100)	42,348,690	34,833,614	25,379,577	22,561,702	19,115,276	17,237,790	16,087,737	15,048,146	14,449,737	13,032,974	3,767,824
CUMULATIVE CASH FLOW	(20,082,900)	(133,386,000)	(91,037,310)	(56,203,696)	(30,824,119)	(8,262,417)	10,852,859	28,090,649	44,178,386	59,226,532	73,676,269	86,709,243	90,477,068
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	13.22
NPV10%	12,990,985
NPV12.5%	2,575,727
NPV15%	(5,735,228)
PAYBACK (YEARS)	4.43
Cu,$US/lb.	1.80
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 1.80/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,300)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%) copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			4.52	4.52	4.52	4.52	4.52	4.52	4.52	4.52	4.52	4.52	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			60,840,202	60,840,202	51,714,171	51,714,171	51,714,171	51,714,171	51,714,171	51,714,171	60,840,202	60,840,202	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			64,906,123	64,906,123	55,170,205	55,170,205	55,170,205	55,170,205	55,170,205	55,170,205	64,906,123	64,906,123	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			1,947,184	1,947,184	1,655,106	1,655,106	1,655,106	1,655,106	1,655,106	1,655,106	1,947,184	1,947,184	
TOTAL OPERATING COSTS			66,857,673	55,253,802	48,017,675	40,891,053	39,330,124	36,993,259	34,975,622	34,108,750	44,043,681	45,729,063	
OPERATING PROFIT			(1,951,550)	9,652,321	7,152,529	14,279,152	15,840,080	18,176,946	20,194,582	21,061,455	20,862,442	19,177,060	
TAXES AT 57%			(1,112,383)	5,501,823	4,076,942	8,139,117	9,028,846	10,360,859	11,510,912	12,005,029	11,891,592	10,930,924	
NET INCOME			(839,166)	4,150,498	3,075,588	6,140,035	6,811,235	7,816,087	8,683,670	9,056,426	8,970,850	8,246,136	
CASH FLOW	(20,082,900)	(113,303,100)	39,176,634	31,661,558	22,683,330	19,865,455	16,419,028	14,541,542	13,391,489	12,351,899	11,277,681	9,860,918	3,767,824
CUMULATIVE CASH FLOW	(20,082,900)	(133,386,000)	(94,209,366)	(62,547,808)	(39,864,479)	(19,999,024)	(3,579,996)	10,961,546	24,353,036	36,704,934	47,982,616	57,843,534	61,611,358
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	9.56
NPV10%	(1,701,818)
NPV12.5%	**********
NPV15%	**********
PAYBACK (YEARS)	5.25
Cu,$US/lb.	1.60
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 1.60/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			3.96	3.96	3.96	3.96	3.96	3.96	3.96	3.96	3.96	3.96	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			53,235,176	53,235,176	45,249,900	45,249,900	45,249,900	45,249,900	45,249,900	45,249,900	53,235,176	53,235,176	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			57,301,098	57,301,098	48,705,933	48,705,933	48,705,933	48,705,933	48,705,933	48,705,933	57,301,098	57,301,098	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			1,719,033	1,719,033	1,461,178	1,461,178	1,461,178	1,461,178	1,461,178	1,461,178	1,719,033	1,719,033	
TOTAL OPERATING COSTS			66,629,522	55,025,651	47,823,747	40,697,124	39,136,196	36,799,331	34,781,694	33,914,821	43,815,530	45,500,912	
OPERATING PROFIT			(9,328,424)	2,275,447	882,186	8,008,809	9,569,737	11,906,602	13,924,239	14,791,112	13,485,568	11,800,186	
TAXES AT 57%			(5,317,202)	1,297,005	502,846	4,565,021	5,454,750	6,786,763	7,936,816	8,430,934	7,686,774	6,726,106	
NET INCOME			(4,011,222)	978,442	379,340	3,443,788	4,114,987	5,119,839	5,987,423	6,360,178	5,798,794	5,074,080	
CASH FLOW	(20,082,900)	(113,303,100)	36,004,578	28,489,502	19,987,082	17,169,207	13,722,781	11,845,294	10,695,242	9,655,651	8,105,625	6,688,862	3,767,824
CUMULATIVE CASH FLOW	(20,082,900)	(133,386,000)	(97,381,422)	(68,891,920)	(48,904,838)	(31,735,631)	(18,012,851)	(6,167,556)	4,527,686	14,183,337	22,288,962	28,977,824	32,745,648
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	5.47
NPV10%	##########
NPV12.5%	##########
NPV15%	##########
PAYBACK (YEARS)	6.58
Cu,$US/lb.	1.40
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 1.40/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			3.39	3.39	3.39	3.39	3.39	3.39	3.39	3.39	3.39	3.39	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			45,630,151	45,630,151	38,785,629	38,785,629	38,785,629	38,785,629	38,785,629	38,785,629	45,630,151	45,630,151	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			49,696,073	49,696,073	42,241,662	42,241,662	42,241,662	42,241,662	42,241,662	42,241,662	49,696,073	49,696,073	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			1,490,882	1,490,882	1,267,250	1,267,250	1,267,250	1,267,250	1,267,250	1,267,250	1,490,882	1,490,882	
TOTAL OPERATING COSTS			66,401,371	54,797,500	47,629,819	40,503,196	38,942,268	36,605,403	34,587,766	33,720,893	43,587,379	45,272,761	
OPERATING PROFIT			(16,705,299)	(5,101,428)	(5,388,157)	1,738,465	3,299,394	5,636,259	7,653,896	8,520,768	6,108,693	4,423,312	
TAXES AT 57%			(9,522,020)	(2,907,814)	(3,071,250)	990,925	1,880,654	3,212,668	4,362,720	4,856,838	3,481,955	2,521,288	
NET INCOME			(7,183,278)	(2,193,614)	(2,316,908)	747,540	1,418,739	2,423,591	3,291,175	3,663,930	2,626,738	1,902,024	
CASH FLOW	(20,082,900)	(113,303,100)	32,832,522	25,317,446	17,290,834	14,472,960	11,026,533	9,149,047	7,998,994	6,959,404	4,933,569	3,516,806	3,767,824
CUMULATIVE CASH FLOW	(20,082,906)	(133,386,000)	(100,553,478)	(75,236,032)	(57,945,198)	(43,472,238)	(32,445,705)	(23,296,659)	(15,297,665)	(8,338,261)	(3,404,692)	112,114	3,879,939
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	0.71
NPV10%	###########
NPV12.5%	###########
NPV15%	###########
PAYBACK (YEARS)	9.97
Cu,$US/lb.	1.20
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 1.20/lb Cu

HAIL HARPER CREEK CASHFLOW..MAY, 1988 - PBK ESTIMATE

	YEAR (MINUS 2)	YEAR (MINUS 1)	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11
CAPITAL EXPENDITURES ($)	(20,082,900)	(113,303,100)		(500,000)									SALVAGE 3,767,824
OVERBURDEN STRIPPING													
Cubic Meters			1,244,300	1,244,300	1,244,300								
Cost ($)			1,244,300	1,244,300	1,244,300								
OPEN PIT PRODUCTION													
Mining strip ratio			1.30	1.40	1.50	1.50	1.50	1.60	1.60	1.70	2.20	2.50	
Waste mined (t)			5,733,000	6,174,000	6,615,000	6,615,000	6,615,000	7,056,000	7,056,000	7,497,000	9,702,000	11,025,000	
Ore mined (t)			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Ore grades													
copper, %			0.400	0.400	0.340	0.340	0.340	0.340	0.340	0.340	0.400	0.400	
gold, g/t			0.046	0.046	0.039	0.039	0.039	0.039	0.039	0.039	0.046	0.046	
silver, g/t			2.514	2.514	2.137	2.137	2.137	2.137	2.137	2.137	2.514	2.514	
MILL PRODUCTION													
Ore milled, t.			4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	4,410,000	
Recovery, %													
copper			87	87	87	87	87	87	87	87	87	87	
gold			75	75	75	75	75	75	75	75	75	75	
silver			75	75	75	75	75	75	75	75	75	75	
Recovery to product													
copper, kg			15,346,800	15,346,800	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	13,044,780	15,346,800	15,346,800	
gold, g			151,200	151,200	128,520	128,520	128,520	128,520	128,520	128,520	151,200	151,200	
silver, g			8,316,000	8,316,000	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	7,068,600	8,316,000	8,316,000	
Concentrate grade, (%)													
copper			28	28	28	28	28	28	28	28	28	28	
Concentrate produced, t													
copper			54810.00	54810.00	46588.50	46588.50	46588.50	46588.50	46588.50	46588.50	54810.00	54810.00	
MINERAL/METAL VALUES($Cdn)													
copper ($/kg)			3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	
gold ($/g)			18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	18.80	
silver, ($/g)			0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28	
SALES REVENUE ($)													
copper			41,067,136	41,067,136	34,907,066	34,907,066	34,907,066	34,907,066	34,907,066	34,907,066	41,067,136	41,067,136	
gold			2,105,038	2,105,038	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	1,789,282	2,105,038	2,105,038	
silver			1,960,883	1,960,883	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,666,751	1,960,883	1,960,883	
TOTAL SALES REVENUE			45,133,057	45,133,057	38,363,099	38,363,099	38,363,099	38,363,099	38,363,099	38,363,099	45,133,057	45,133,057	
OPERATING COSTS ($)													
mine			10,464,287	10,865,156	12,324,425	11,080,125	13,636,823	14,182,295	14,182,295	14,727,768	25,359,264	27,736,695	
mill			12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	12,116,475	
plant services			1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	1,457,946	
administration			855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	855,981	
depreciation			40,015,800	28,011,060	19,607,742	13,725,419	9,607,794	6,725,456	4,707,819	3,295,473	2,306,831	1,614,782	
royalty			1,353,992	1,353,992	1,150,893	1,150,893	1,150,893	1,150,893	1,150,893	1,150,893	1,353,992	1,353,992	
TOTAL OPERATING COSTS			66,264,481	54,660,610	47,513,462	40,386,839	38,825,911	36,489,046	34,471,409	33,604,536	43,450,489	45,135,871	
OPERATING PROFIT			(21,131,423)	(9,527,552)	(9,150,363)	(2,023,741)	(462,812)	1,874,053	3,891,690	4,758,562	1,682,569	(2,813)	
TAXES AT 57%			(12,044,911)	(5,430,705)	(5,215,707)	(1,153,532)	(263,803)	1,068,210	2,218,263	2,712,381	959,064	(1,603)	
NET INCOME			(9,086,512)	(4,096,847)	(3,934,656)	(870,208)	(199,009)	805,843	1,673,427	2,046,182	723,504	(1,210)	
CASH FLOW	(20,082,900)	(113,303,100)	30,929,288	23,414,213	15,673,086	12,855,211	9,408,784	7,531,298	6,381,245	5,341,655	3,030,336	1,613,572	3,767,824
CUMULATIVE CASH FLOW	(20,082,900)	(133,386,000)	(102,456,712)	(79,042,499)	(63,369,414)	(50,514,203)	(41,105,418)	(33,574,120)	(27,192,875)	(21,851,220)	(18,820,884)	(17,207,312)	(13,439,487)
DEPRECIATED VALUE			93,370,200	65,359,140	45,751,398	32,025,979	22,418,185	15,692,730	10,984,911	7,689,437	5,382,606	3,767,824	

DCF-ROI	-2.67
NPV10%	##########
NPV12.5%	##########
NPV15%	##########
PAYBACK (YEARS)	0.00
Cu,$US/lb.	1.08
Au,$US/oz	456.00
Ag,$US/oz	6.69

OPERATING COST	$/t
MINE, YEAR 1,2	0.91
YEAR 3,4	1.01
YEAR 5-8	1.24
YEAR 9,10	1.80
AVERAGE	1.24
CONCENTRATOR	2.75
SERVICES	0.33
ADMINISTRATION	0.19

NEW EQUIPMENT; $US 1.08/lb Cu

8.0 FUTURE WORK PROGRAM

8.1 Further Geological Development

Mineralizing hydrothermal fluids are very likely responsible for the copper mineralization in the Hail Harper Creek Area.

The pattern of fluid circulation is not as well outlined in the East Zone as it is in the West Zone.

As a consequence, mineralization in the East Zone is more diffuse compared to that of the West Zone where mineralization follows discrete channels with a better discrimination between Cu grade values. In the West Zone, ore bodies are more clearly outlined and continuity of ore grade is more consistent.

In the West Zone, a drilling interval of 100 to 120 metres is probably sufficient. More drilling will only be necessary to fill the holes in the grid and confirm the northern down dipping extension of the north mineralized zone to a depth corresponding to level 1500 metres.

No. of holes: 15 - Total metrage: 2570 m.

In the East Zone, even though the amount of drilling is almost the same as it is in the West Zone (± 70 holes in E. Zone compared to ± 90 holes in W. Zone), the more widespread distribution of the ore results in an insufficient drilling coverage.

Average distance between main drilling sections:

in W. Zone: 85 m
in E. Zone: 105 m

The irregularity of grade distribution in the East Zone would imply a shorter interval between drill holes. No mathematical or statistical computations were operated on East Zone data, however it is felt that an interval of 90 metres should be a maximum.

It appears obvious from observations of the sections that the mineralization in the East Zone shows a down dip extension towards the north with a definite increase of the copper grade and a less diffuse overall aspect (more related to the West Zone type of mineralization).

More drilling to the north of the East Zone is then recommended to fill the holes in the grid down to the 1500 metre level and even deeper to confirm the possible extension to the north of relatively richer and more consistent mineralization.

Filling of the grid in and between main sections:
No. of holes: 12 - total metrage: 2190 m

Filling of the grid to the north down to level 1500 m:
No. of holes: 7 - total metrage: 1580 m

It is also recommended to verify the possible extension at depth of the mineralization toward the north to confirm the apparent increase of grade in that direction.

East Zone
No. of holes: 15 - total metrage: 4960 m

West Zone
No. of holes: 8 - total metrage: 2600 m

8.2 Mine Planning

8.2.1 Geotechnical Program

The next step is to conduct a thorough geotechnical study in order to determine the optimum slopes for the different rock types, permitting a more detailed pit design to be produced. This will involve the physical testing of representative samples in a laboratory.

8.2.2 Optimum Pit Design

Based on these new parameters, from 8.2.1 the revised geological interpretation resulting from the drilling program detailed in 8.1, a revised ore distribution and, computer program, an optimum pit design should be produced.

8.2.3 Mine planning alternative

Some work should be done to investigate a different approach to materials handling design involving the use of either a cable belt conveyor or a bucket tramway system to transport the ore down the mountain to the valley. The shape of the orebody, its dip towards the north, and its apparent increase in copper grade with depth, all make these, and possibly other alternatives, worth considering. Both of these systems are well proven in the mining industry. Examples of past applications which were very successful are Craigmont Mines, a former copper producer near Merritt, B.C., the Black Angel Mine in Greenland and the Monarch Mine near Field, B.C.

One of the most significant consequences of being able to bring the ore down to the valley economically, would be the cost savings from both a capital and an operating cost point of view of having the plant in the valley bottom, rather than up the mountain. One potential problem would be the disposal of tailings, but the extra cost which might be involved in treating them so they would be acceptable for the valley floor, might be insignificant compared to the cost of the plant being up the mountain.

8.3 Process

Alternatives Considered and Suggestions for Future Work

8.3.1 Recovery of Molybdenum, Titanium and Nickel

During the progress of the present study, PBK have evaluated the opportunities for additional revenues for the project with regard to the recovery of additional products.

8.3.1.1 Molybdenum

While reported analyses for composite samples used in metallurgical testing indicated a molybdenum content of about 0.016% - composite samples of drill cores obtained from Noranda averaged only 0.0015%. While the origin of the discrepancy is not clear, it has been suggested by G. Windler of Noranda that contamination from Cr/Mo steel pulverizer plates may have contributed.

At the indicated lower level in drill core, molybdenum concentrate production would not be economic. However, in any future geological program, its occurrence should be closely followed.

8.3.1.2 Titanium

Limited indications of levels of about 1.75% titanium oxide, and some geological reference to the presence of rutile spurred initial interest in this mineral. In the course of the study, however, it became evident that the predominant titanium mineral is sphene, 41 % TiO_2 versus 100 % TiO_2 for rutile. Due to the low titanium level and processing difficulties, no commercial operations using a sphene feedstock are known.

8.3.1.3 Nickel/Cobalt

Analysis indicates levels of 0.03% Ni and 0.0036% Cobalt which, depending on mineralization, could become economic by-products. Geological evaluation, however, indicates significant pyrrhotite occurrences and makes no mention of specific nickel and cobalt minerals. It is most probable that nickel and cobalt values are present as part of the pyrrhotite matrix, and hence do not represent economic mineralization.

8.3.2 Covered Coarse Ore Stockpile

In order to minimize capital costs and hence early front end expense, Coarse Ore Storage on an open air stockpile has been assumed. Removal of fines direct to fine ore storage has been incorporated to minimize freezing problems in winter.

Due to the altitude of the plant-site and heavy snowfall in the area, a careful evaluation of covered storage at the definitive feasibility stage is recommended. A preliminary estimate of additional capital cost is $2.0 million.

8.3.3 Autogenous Grinding

Increasingly autogenous or semi-autogenous grinding is incorporated into large milling flowsheets. Due to the relative softness of the Hail Harper ore and its friable nature, substantiating laboratory and pilot testing would be required prior to its application. Estimates of impact on capital and operating costs indicate about a 5% capital saving at the expense of 2 to 3% increase in operating cost. Due to the impact on front end expenditure, significant early cash flow benefits generally result.

Preliminary laboratory work is recommended on this approach at the feasibility stage.

8.3.4 Additional Metallurgical Testing

This preliminary feasibility evaluation is based entirely on a limited laboratory test program on two samples of Noranda ore. Additional optimization testing for grind, flotation, and reagents is recommended at the feasibility stage. Work should be carried out on carefully selected representative composits from both the Aurun and Noranda orebodies.

8.3.5 Concentrate Pipelining

It is suggested that the economics of pipelining of concentrate slurry to the bottom of the mountain at Vavenby and installation of filtration and loadout at that point be evaluated at the definitive feasibility stage in order to eliminate trucking costs. The economics of pressure filter dewatering (e.g. Larox, Lasta) in place of the simple low cost disc filter assumed for the present study should also be assessed at that stage.

8.4 Infrastructure

8.4.1 Water supply

A source of water should be sought, either in a natural state, or in the form of a reservoir to be built, the cost of which should be weighed against the cost of getting water up to the plant site from the North Thompson River.

8.4.2 Tailings pond

The technique and cost of cycloning which could be used to raise the dam after the starter dam construction period should be investigated. The cost of the starter dam construction should also be compared to alternate, and possibly less expensive methods.

8.4.3 Power

Part of the power supply might be cheaper if it was diesel, rather than electric power, so this should be checked out. The cost of relatively simple items, such as the heating of the concentrator, could be reduced by a significant amount through the use of natural gas.

8.4.4 Acid leachate

Tests on the waste rock should be done to establish whether this rock does, in fact, present a threat to the environment in its reaction to weathering.

8.4.5 Plant site foundation and levelling

Tests on the overburden and waste rock should be done to confirm its suitability as a means of levelling the proposed plant site. Finding an area large and flat enough to contain the plant layout was not easy, and to keep costs down as much as possible, it was assumed that this material would be adequate, but this must be checked. Alternatively, through the use of more detailed topographical maps and a thorough review, it may be possible to find enough room with less site preparation.

GEOLOGICAL EVALUATION REPORT

PRE-FEASIBILITY STUDY
Drawings

HAIL - HARPER CREEK COPPER PROSPECT

82 M 12 (W½)

for

AURUN MINES LTD

17710 104th Avenue
Surrey, British Columbia, V3R 1R1

by

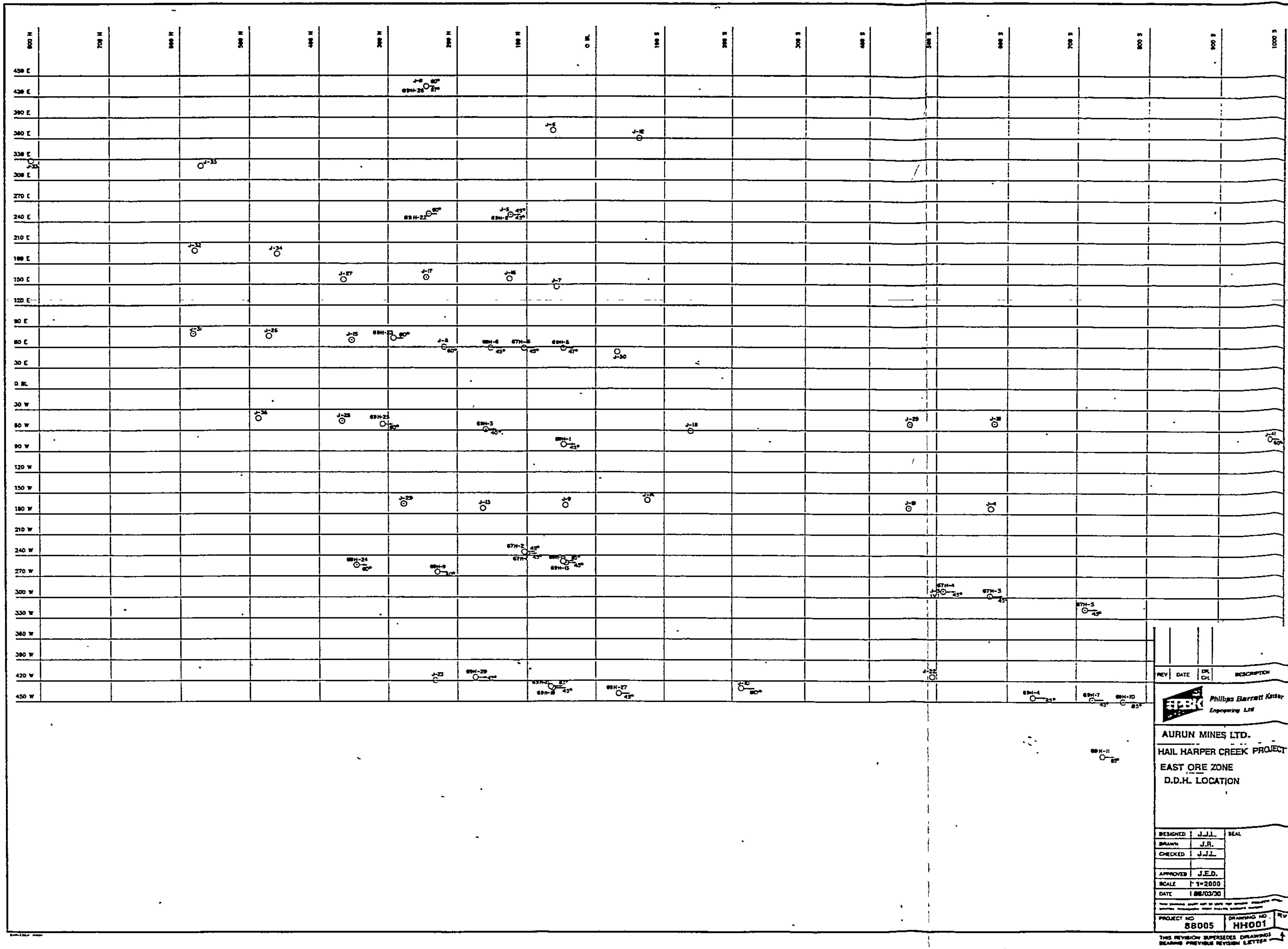
PHILLIPS BARRATT KAISER ENGINEERING LTD

2150 West Broadway
Vancouver, British Columbia, V6K 4L9

May 1988

DRAWING INDEX

GEOLOGY AND MINING - PLANS AND SECTIONS



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100 S
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120 W
150 W
180 W
210 W
240 W
270 W
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360 W
390 W
420 W
450 W
REV
DATE
DR. CH.
DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
D.D.H. LOCATION
DESIGNED J.J.L.
DRAWN J.R.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1=2000
DATE 88/03/30
SEAL
PROJECT NO 88005
DRAWING NO HH001
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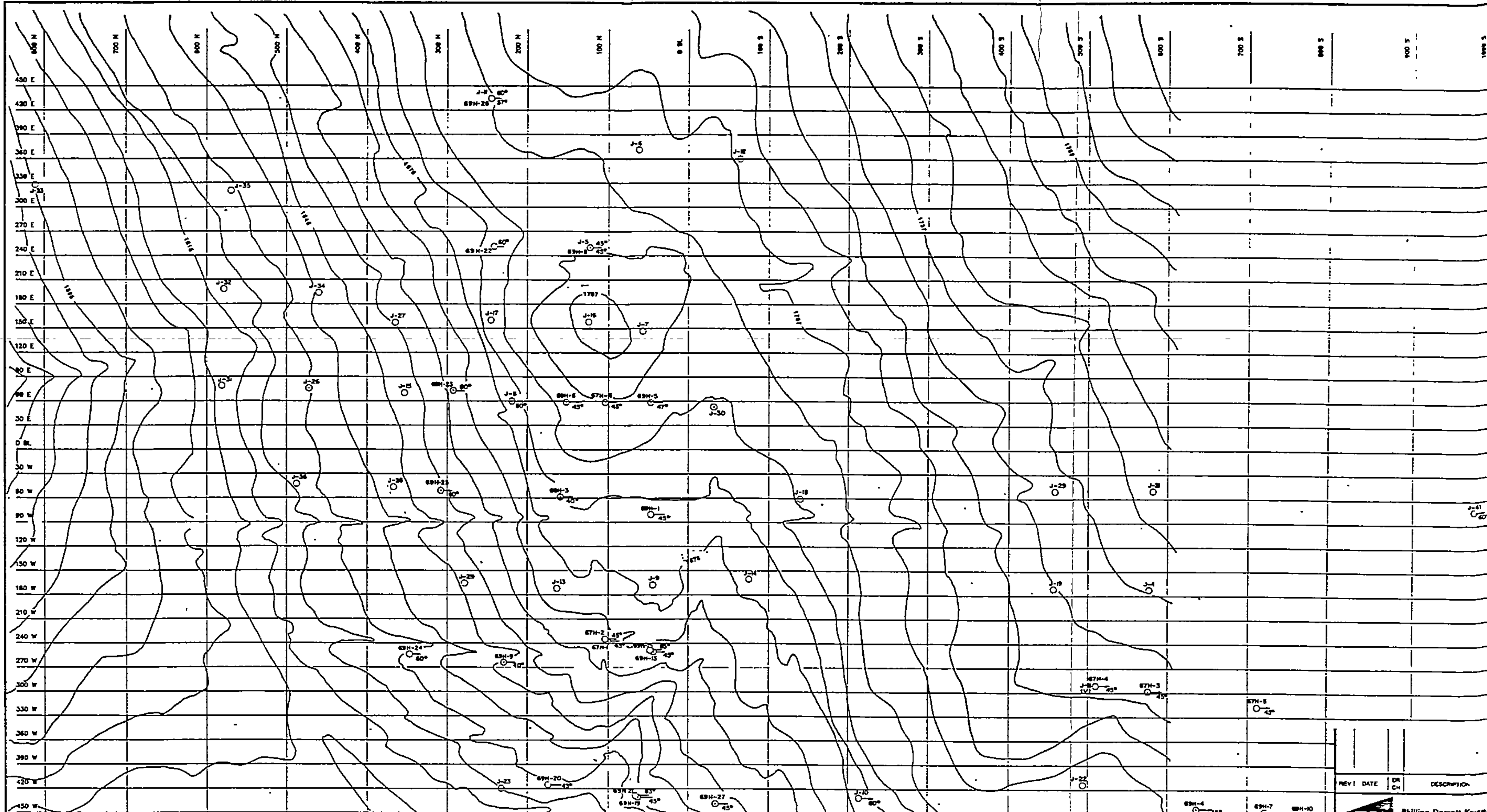

REV	DATE	DR / CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SURFACE PLAN

DESIGNED	J.J.L.	SEAL
DRAWN	J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH002	

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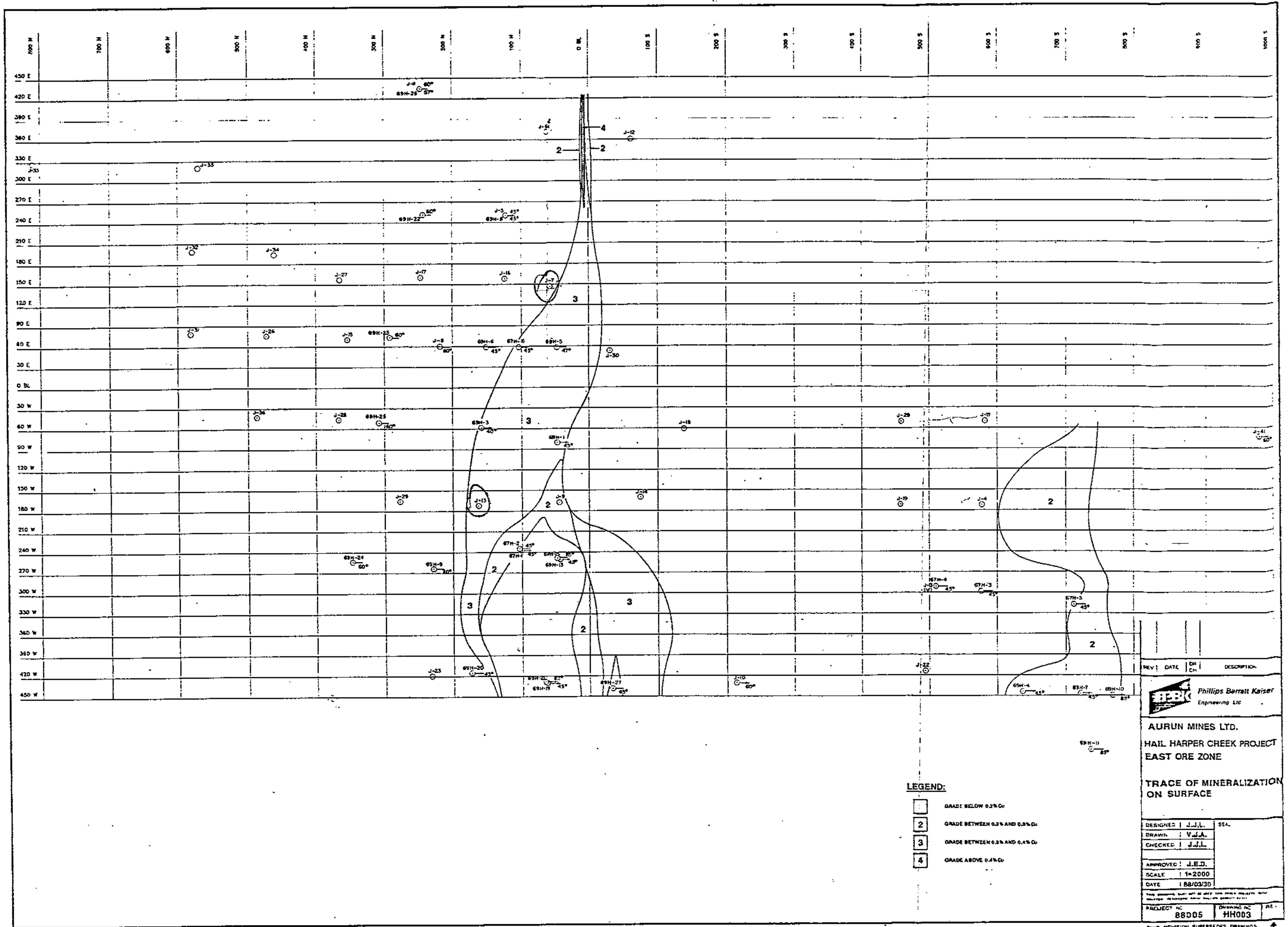
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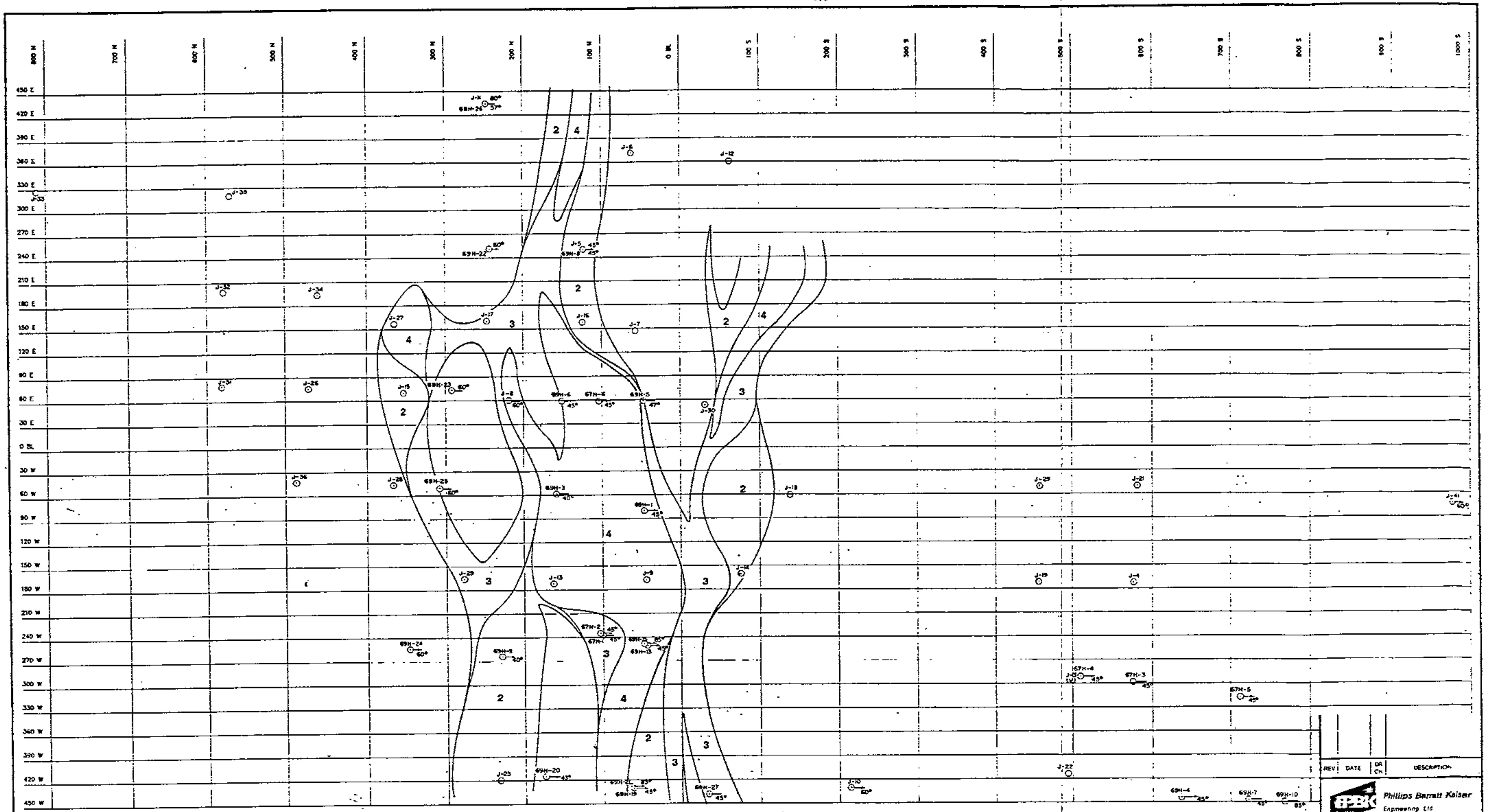
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV | DATE | DR CH | DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
TRACE OF MINERALIZATION
ON SURFACE
DESIGNED | J.J.L.
DRAWN | V.J.A.
CHECKED | J.J.L.
APPROVED | J.E.D.
SCALE | 1=2000
DATE | 88/03/30
SEAL
PROJECT NO 88005
DRAWING NO HH003
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



LEGEND:

	GRADE BELOW 0.2% Cu
2	GRADE BETWEEN 0.2% AND 0.3% Cu
3	GRADE BETWEEN 0.3% AND 0.4% Cu
4	GRADE ABOVE 0.4% Cu

REV	DATE	DR CK	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

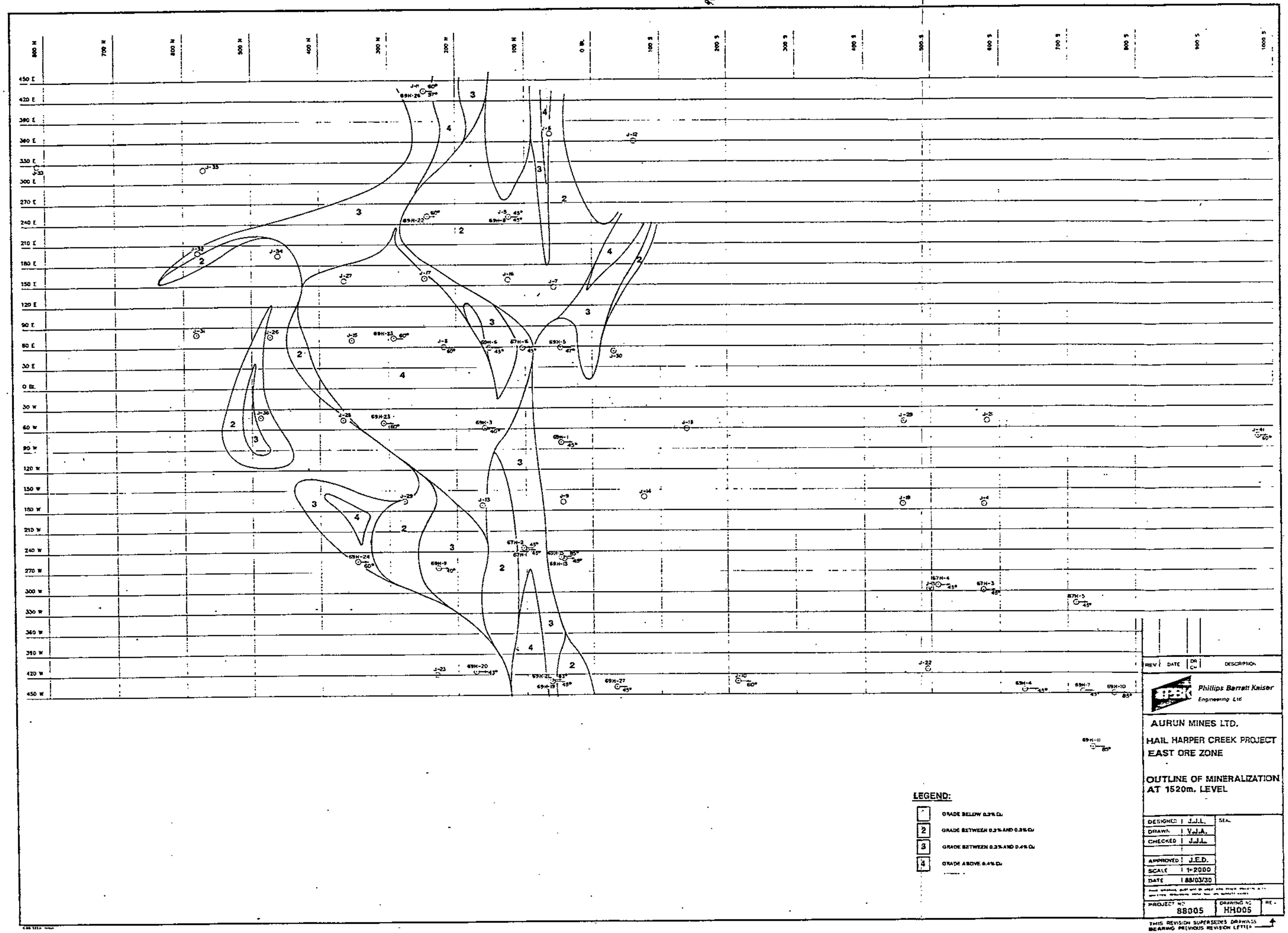
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

OUTLINE OF MINERALIZATION
AT 1600m. LEVEL

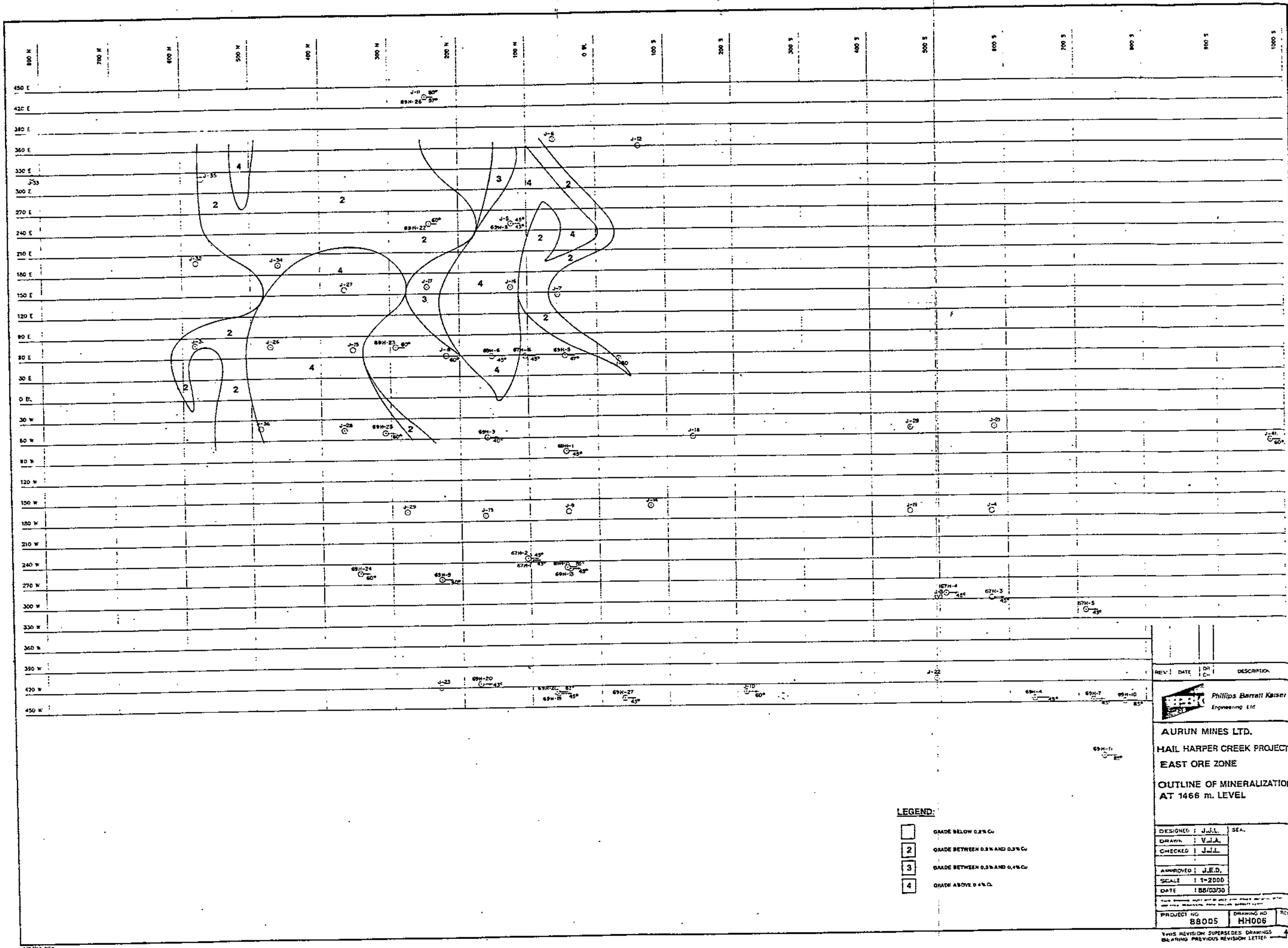
DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH004	

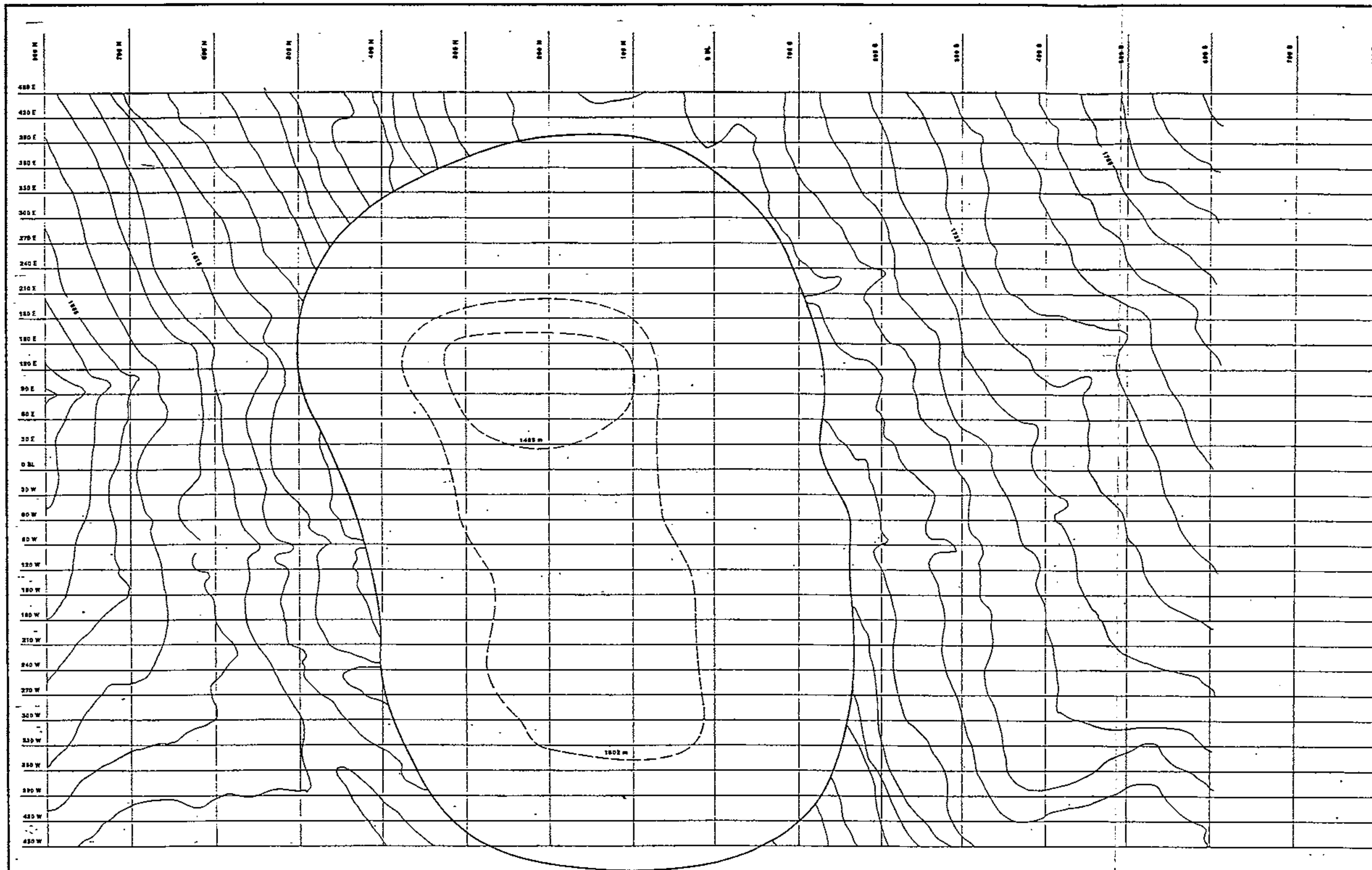
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



LEGEND:
GRADE BELOW 0.2% Cu
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV DATE DR CH DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
OUTLINE OF MINERALIZATION AT 1520m. LEVEL
DESIGNED J.J.L.
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1:2000
DATE 88/03/30
SEAL
PROJECT NO 88005
DRAWING NO HH005
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



LEGEND:
GRADE BELOW 0.2% Cu
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV DATE DR CH DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
OUTLINE OF MINERALIZATION
AT 1466 m. LEVEL
DESIGNED J.J.L.
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1=2000
DATE 88/03/30
SEAL
PROJECT NO 88005
DRAWING NO HH006
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR. CH.	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd.

AURUN MINES LTD.

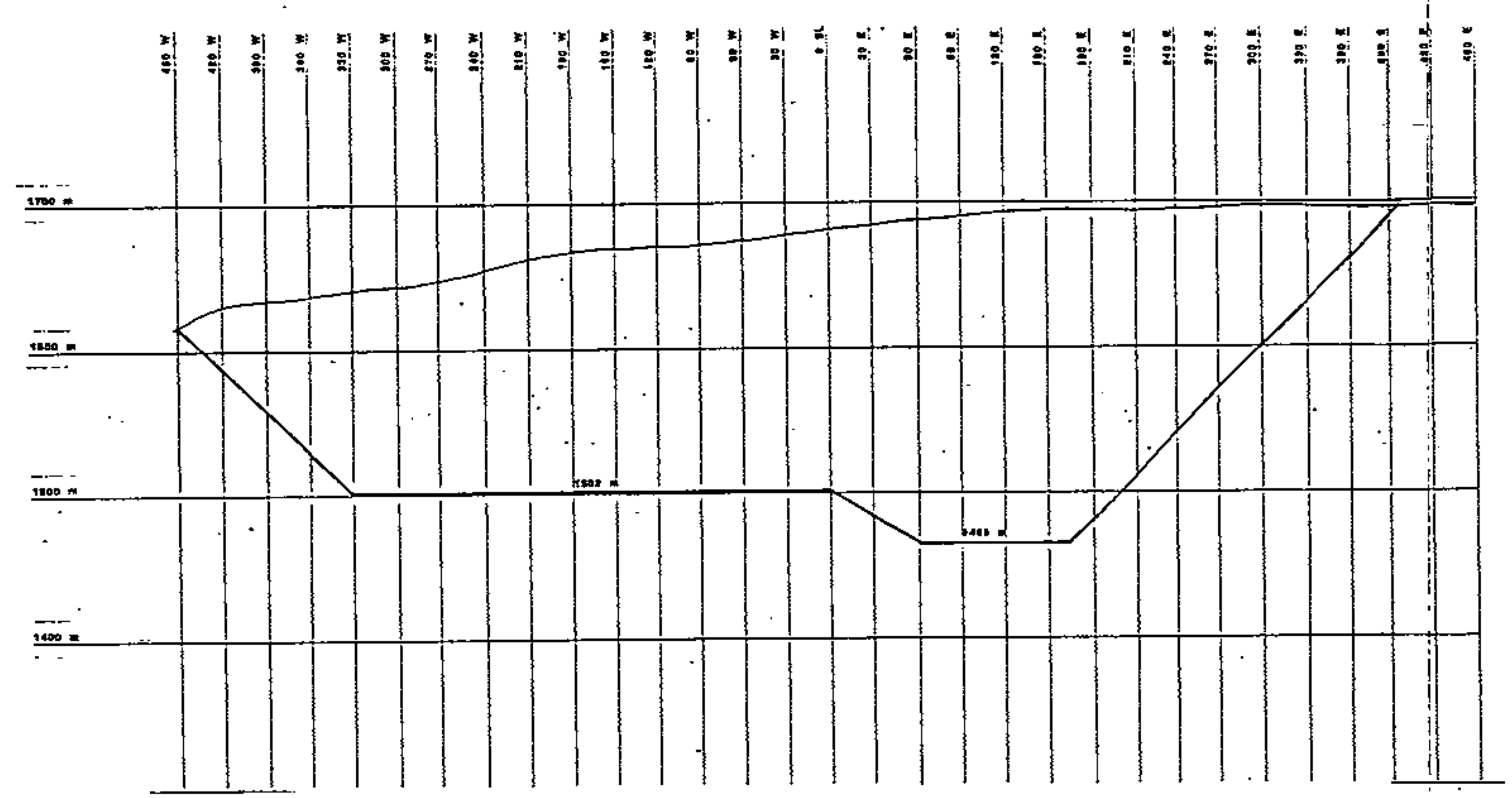
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

FINAL PIT LIMITS
ON SURFACE

DESIGNED	J.K.R.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH007	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR. CH.	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

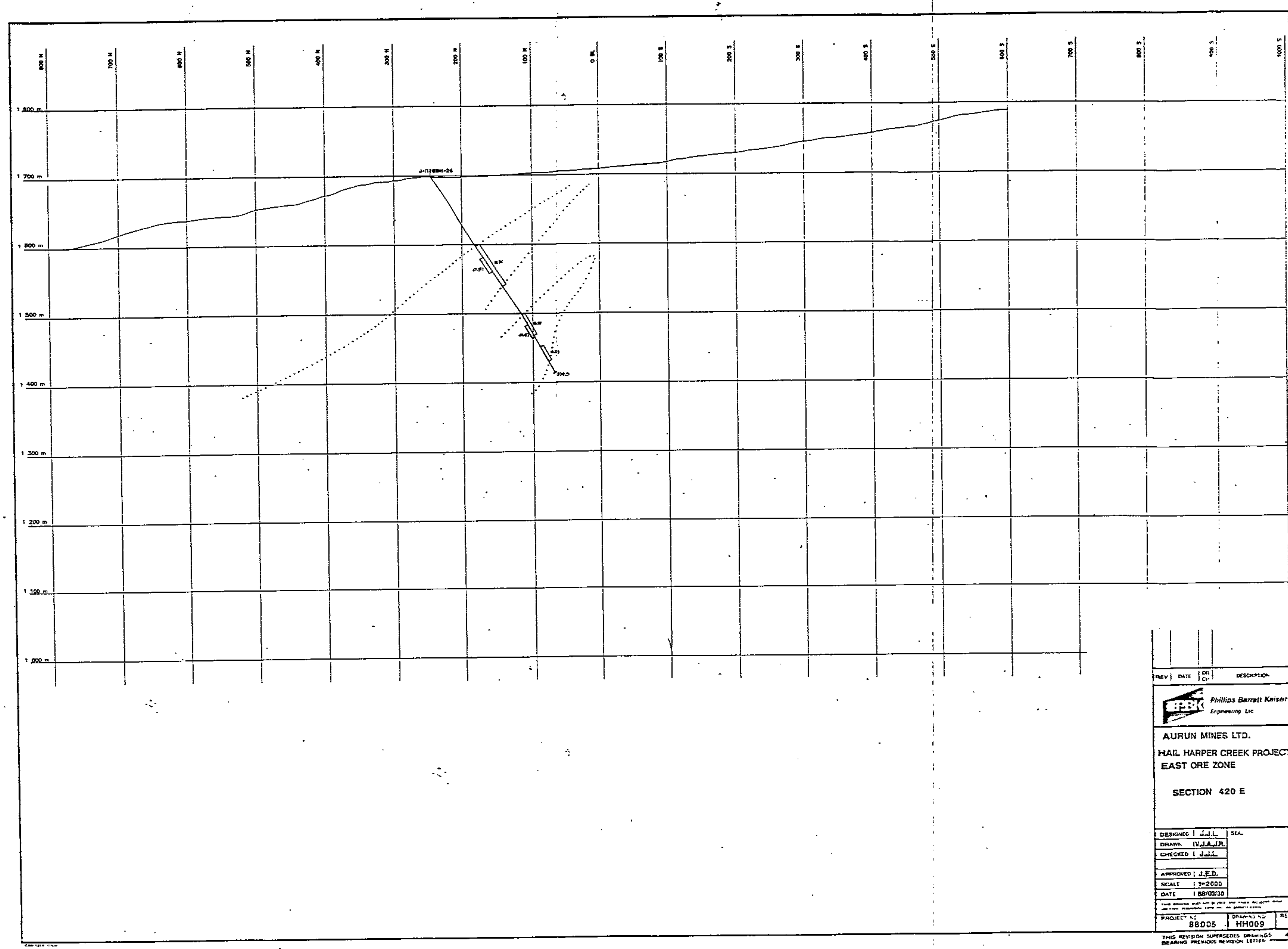
EAST ORE ZONE

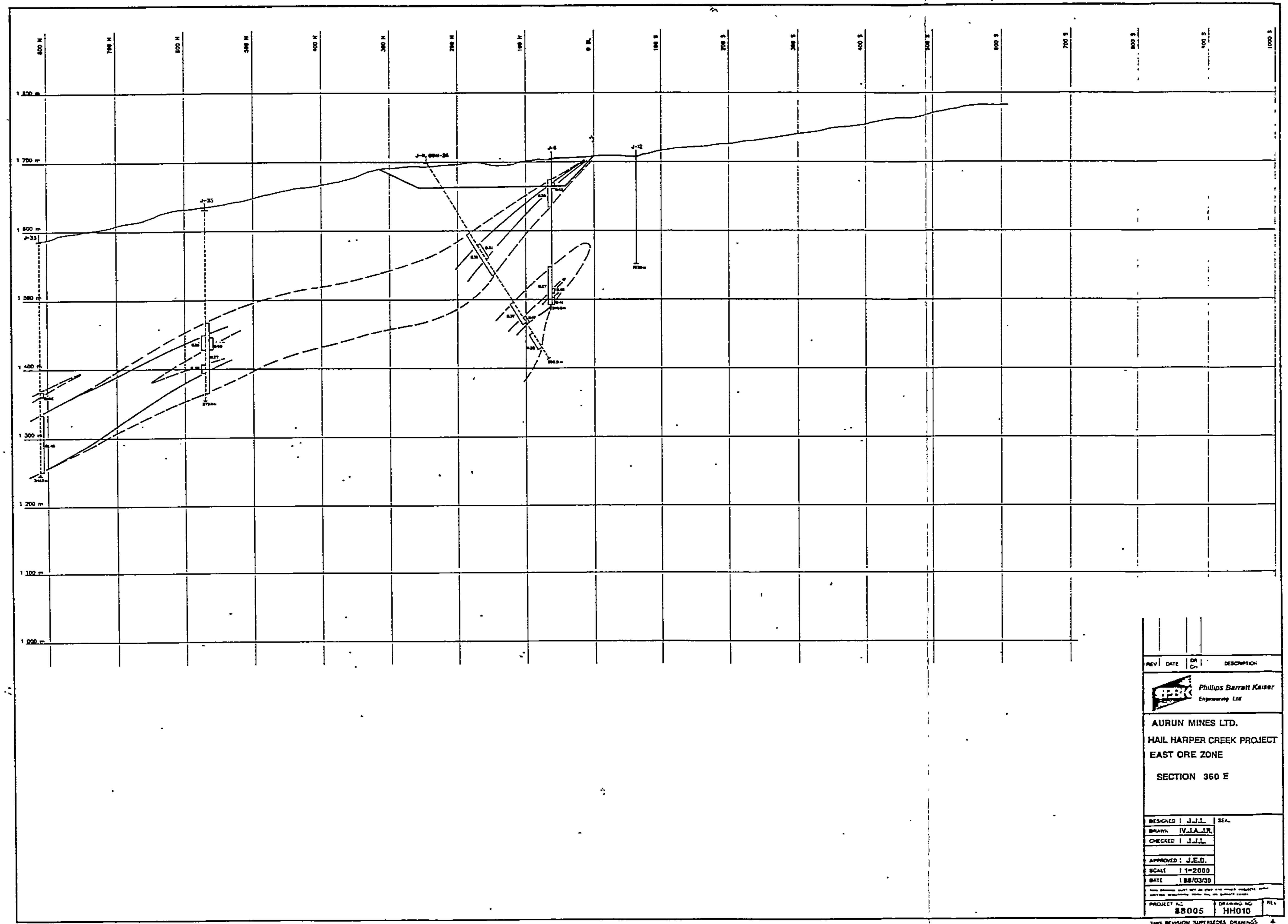
FINAL PIT BOTTOM PROFILE

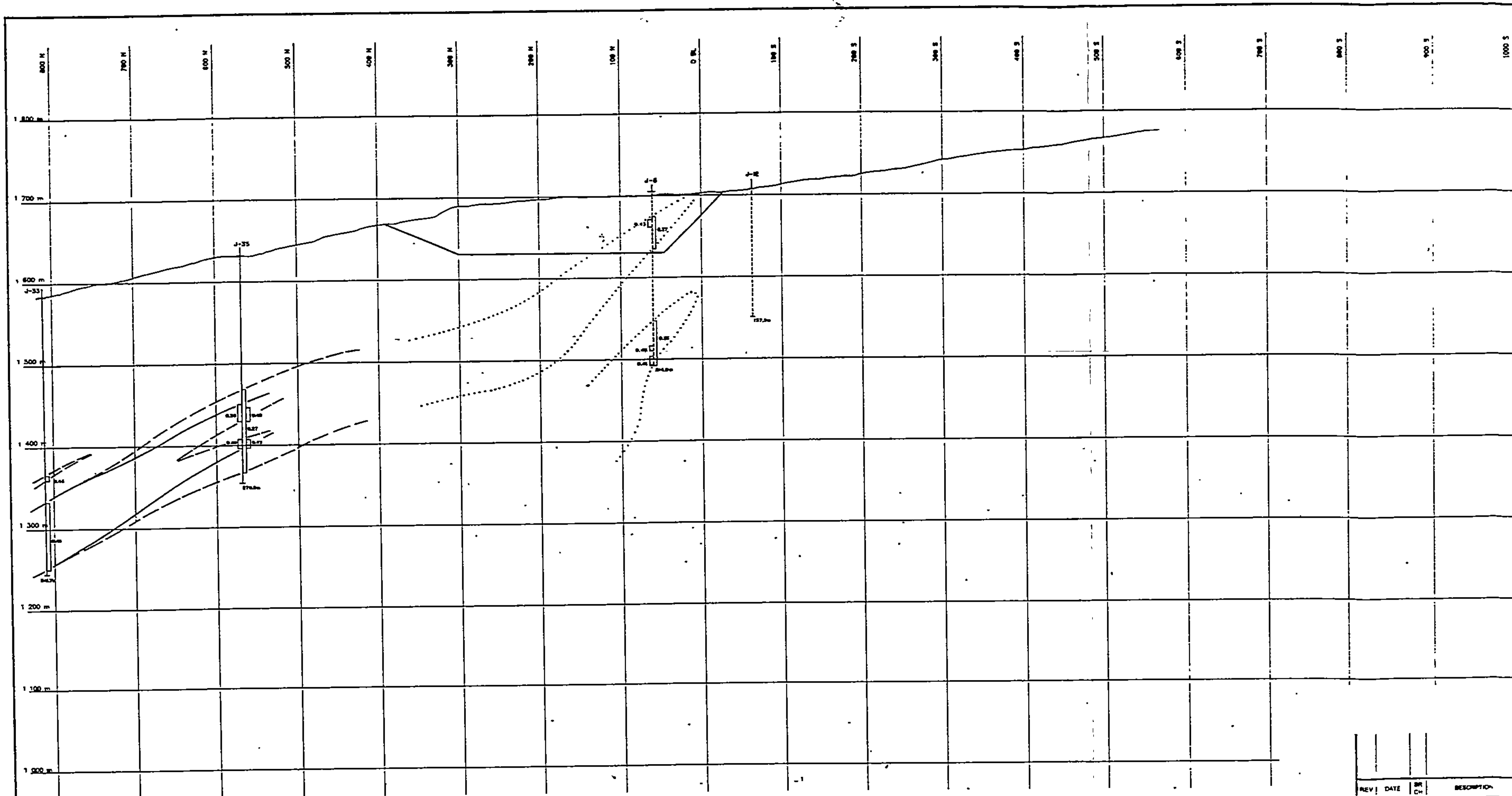
DESIGNED	J.K.R.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH008	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


1 800 m
1 700 m
1 600 m
1 500 m
1 400 m
1 300 m
1 200 m
1 100 m
1 000 m
0.91
0.47
REV
DATE
DR
CH
DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SECTION 420 E
DESIGNED J.J.L.
DRAWN V.J.A.J.R.
CHECKED J.J.L.
SEAL
APPROVED J.E.D.
SCALE 1=2000
PROJECT NO 88005
DRAWING NO HH009
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


800 N
700 N
600 N
500 N
400 N
300 N
200 N
100 N
0 BL
100 S
200 S
300 S
400 S
500 S
600 S
700 S
800 S
900 S
1000 S
1 800 m
1 700 m
1 600 m
1 500 m
1 400 m
1 300 m
1 200 m
1 100 m
1 000 m
J-33
J-35
J-6
J-12
REV
DATE
DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SECTION 360 E
DESIGNED : J.J.L.
DRAWN IV.J.A.J.R.
CHECKED : J.J.L.
SEAL
APPROVED : J.E.D.
SCALE : 1=2000
DATE : 88/03/30
PROJECT NO
88005
DRAWING NO
HH010
REV
THIS REVISION SUPERSEDES DRAWINGS
BEARING PREVIOUS REVISION LETTER



REV	DATE	BR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

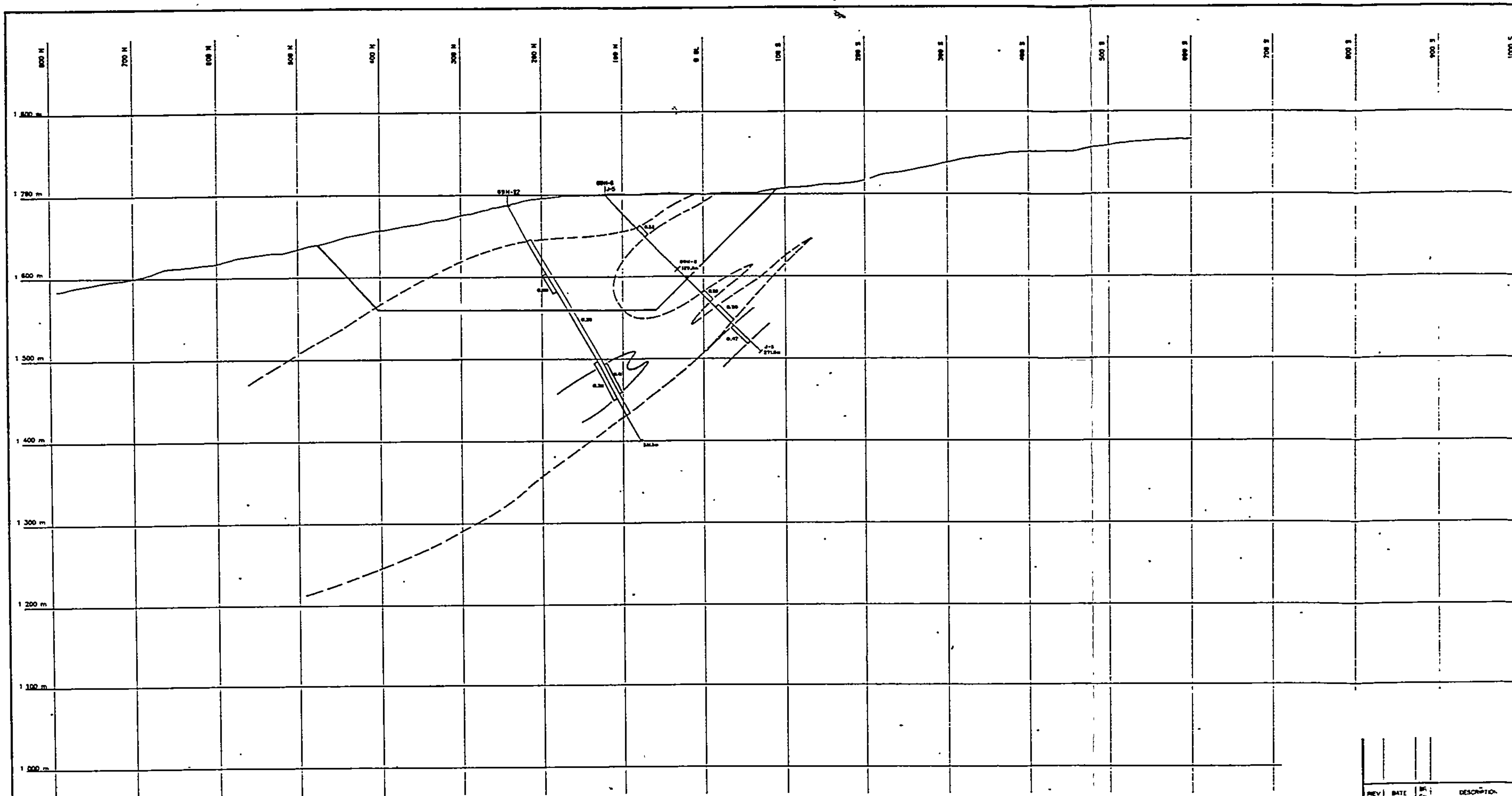
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 330 E

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV
88005	HH011	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER 4



REV	DATE	BY CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

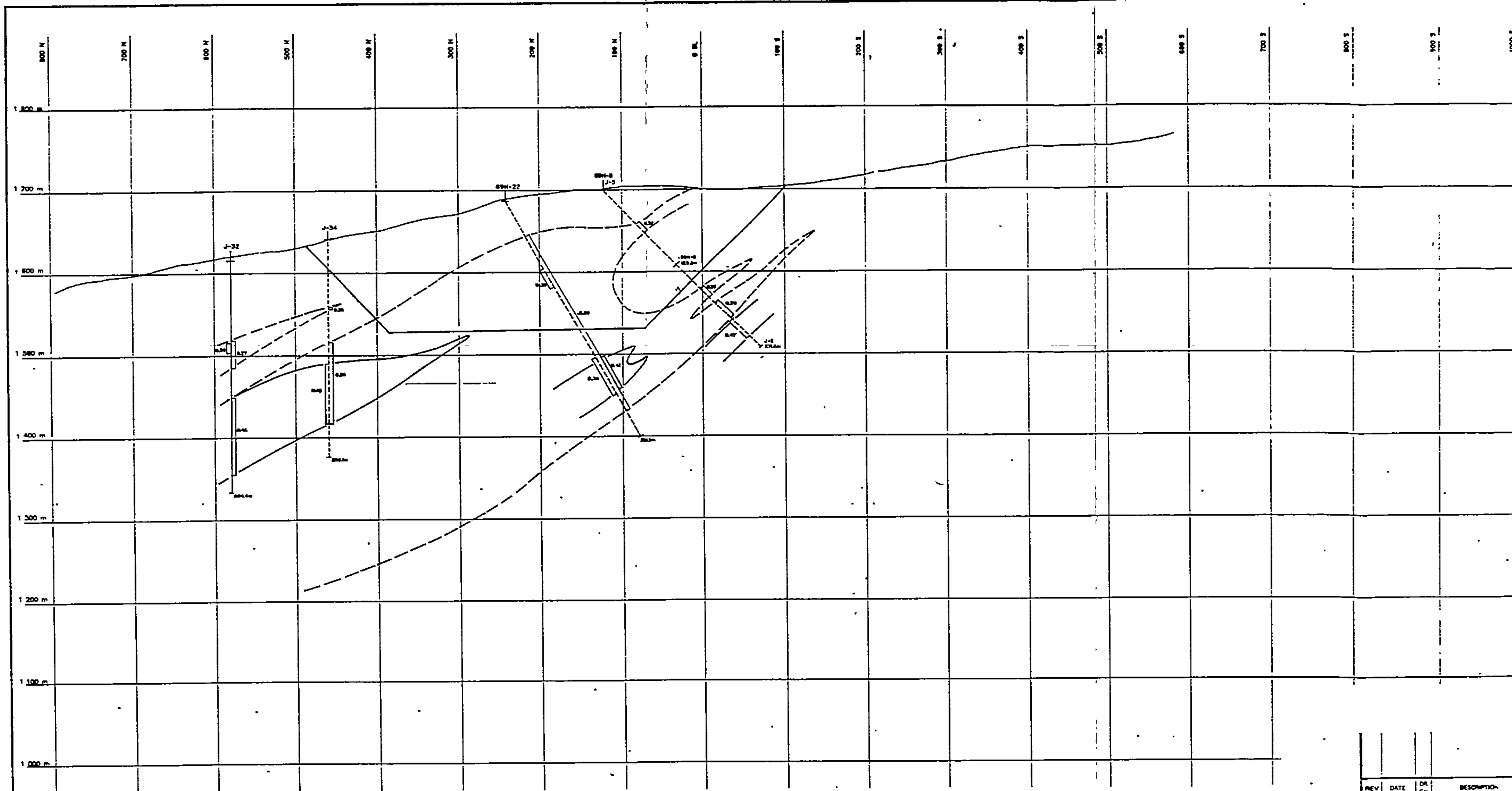
EAST ORE ZONE

SECTION 240 E

DESIGNED	J.J.L.	SEAL
DRAWN	IV.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH012	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

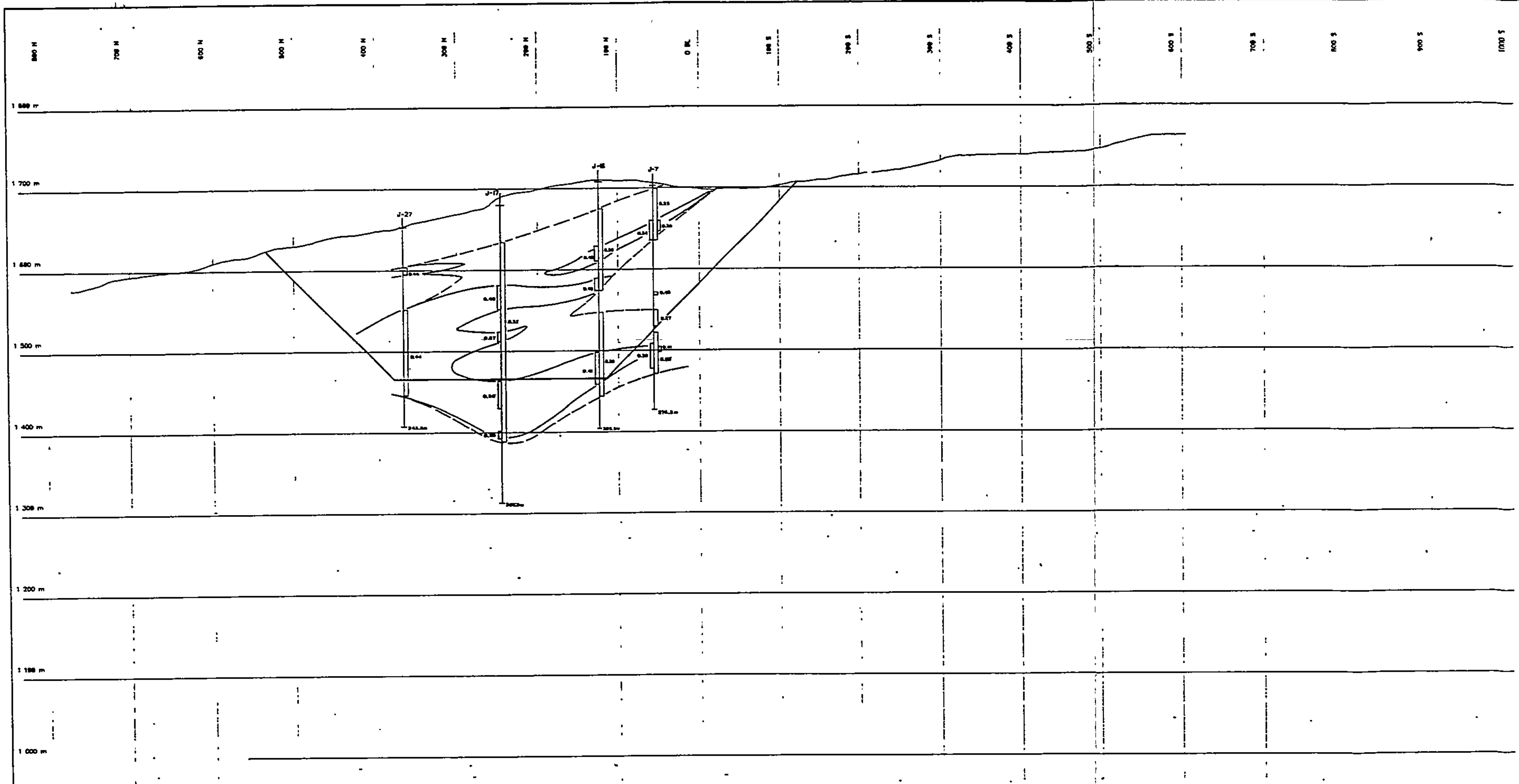
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 210 E

DESIGNED	J.J.L.	SEAL
DRAWN	IV.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No.	DRAWING NO.	REV.
88005	HH013	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



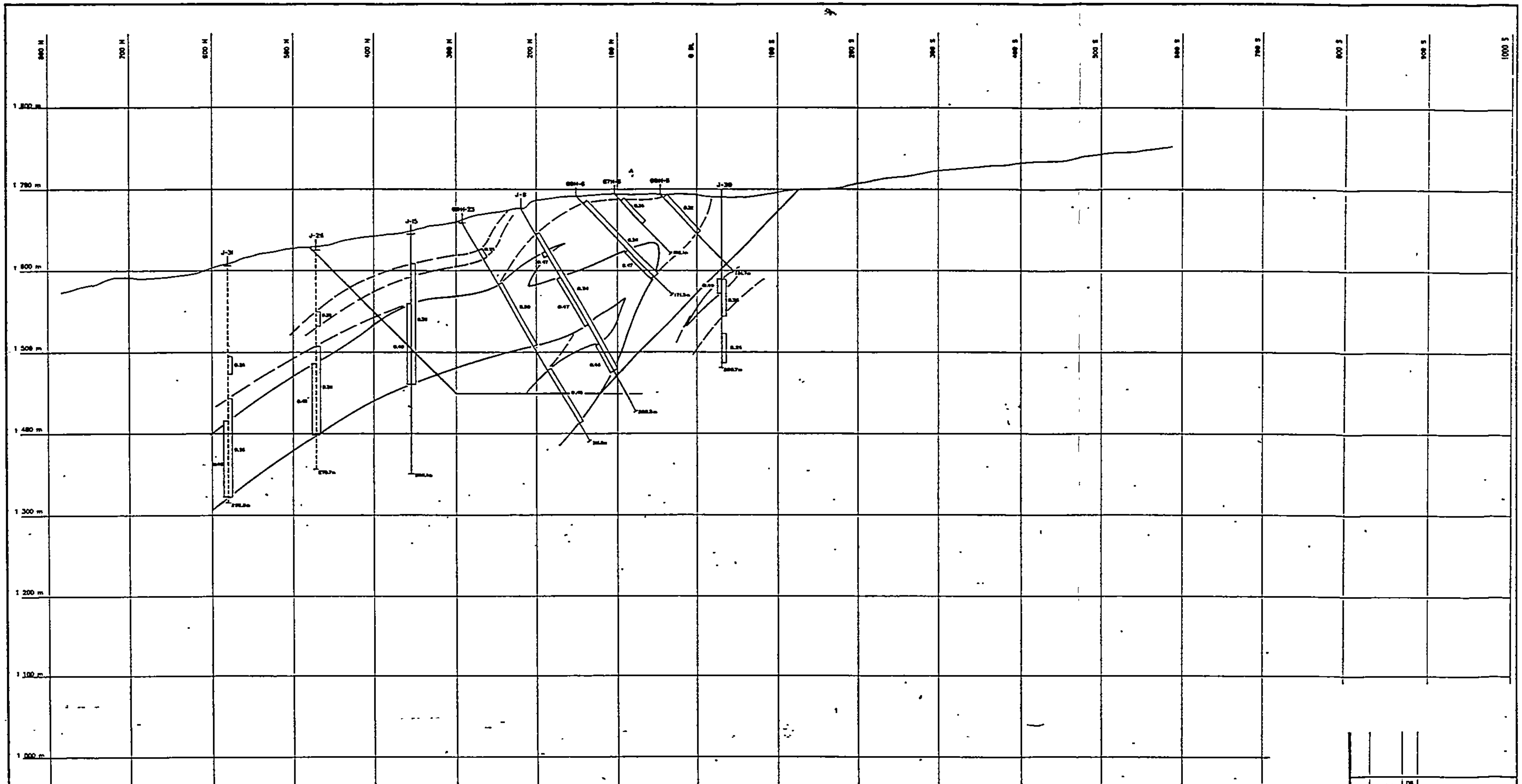
AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 150 E

DESIGNED	J.J.L.
DRAWN	N.J.A.J.R.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	I1=2000
DATE	188/03/30

88005 | HH014



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

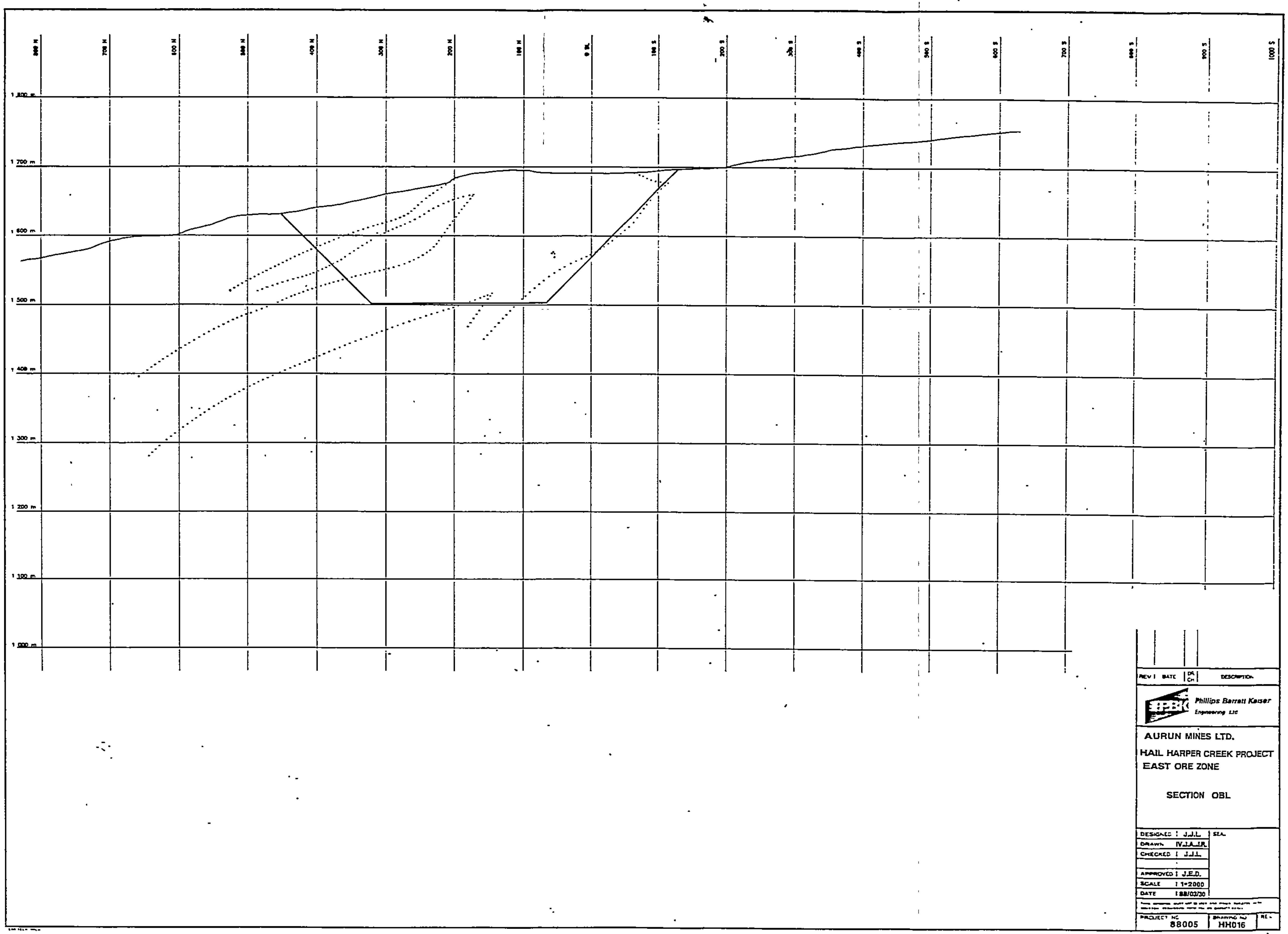
EAST ORE ZONE

SECTION 60 E

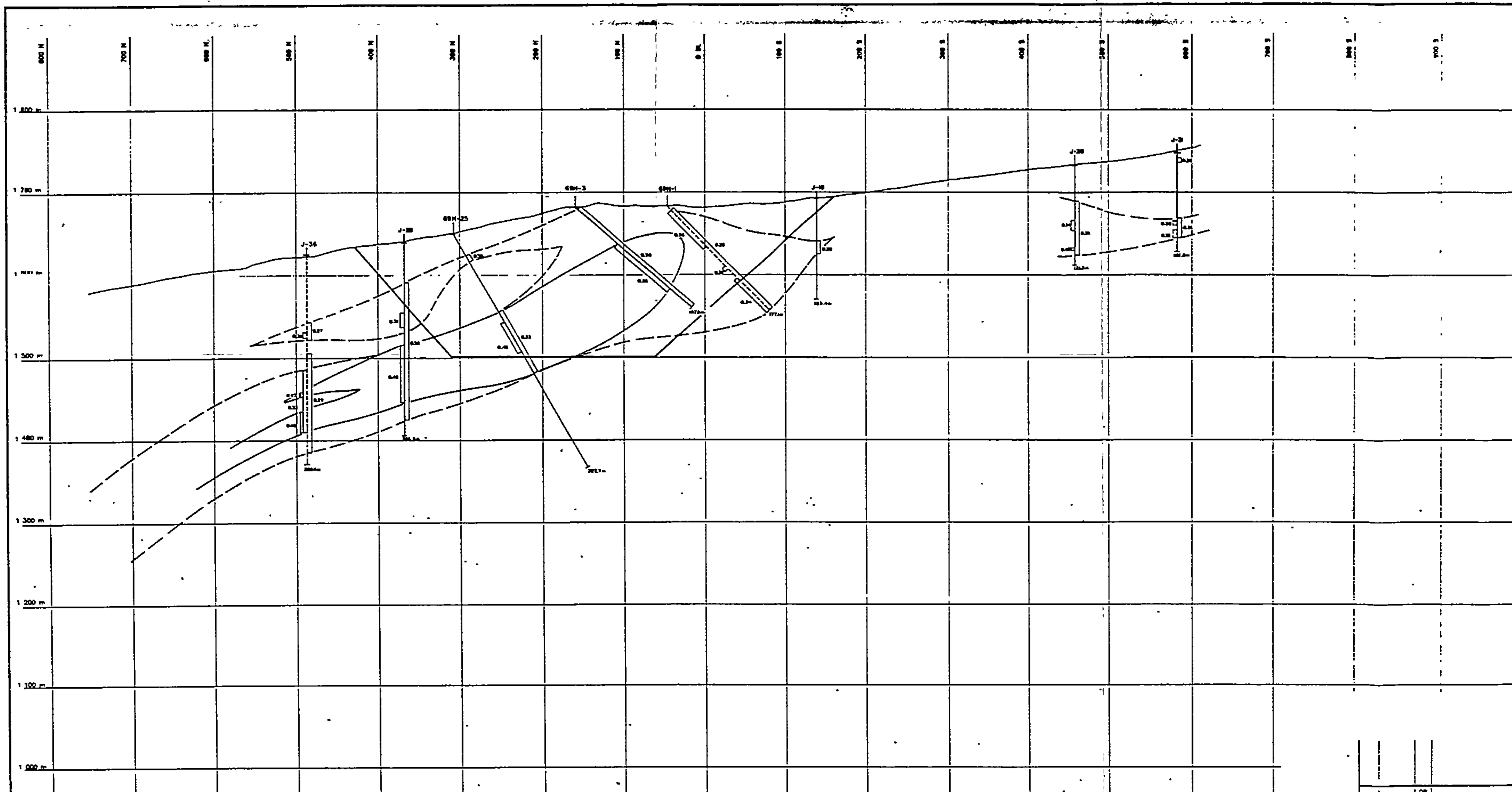
DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH015	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



700 N
400 N
300 N
200 N
100 N
0 BL
100 S
200 S
400 S
500 S
600 S
700 S
900 S
1000 S
1 800 m
1 700 m
1 600 m
1 500 m
1 400 m
1 300 m
1 200 m
1 100 m
1 000 m
REV
DATE
DR CH
DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SECTION OBL
DESIGNED : J.J.L.
SEAL
DRAWN IV.J.A.J.R.
CHECKED : J.J.L.
APPROVED : J.E.D.
SCALE : 1=2000
DATE : 88/03/30
PROJECT NO 88005
DRAWING NO HH016
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR / CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

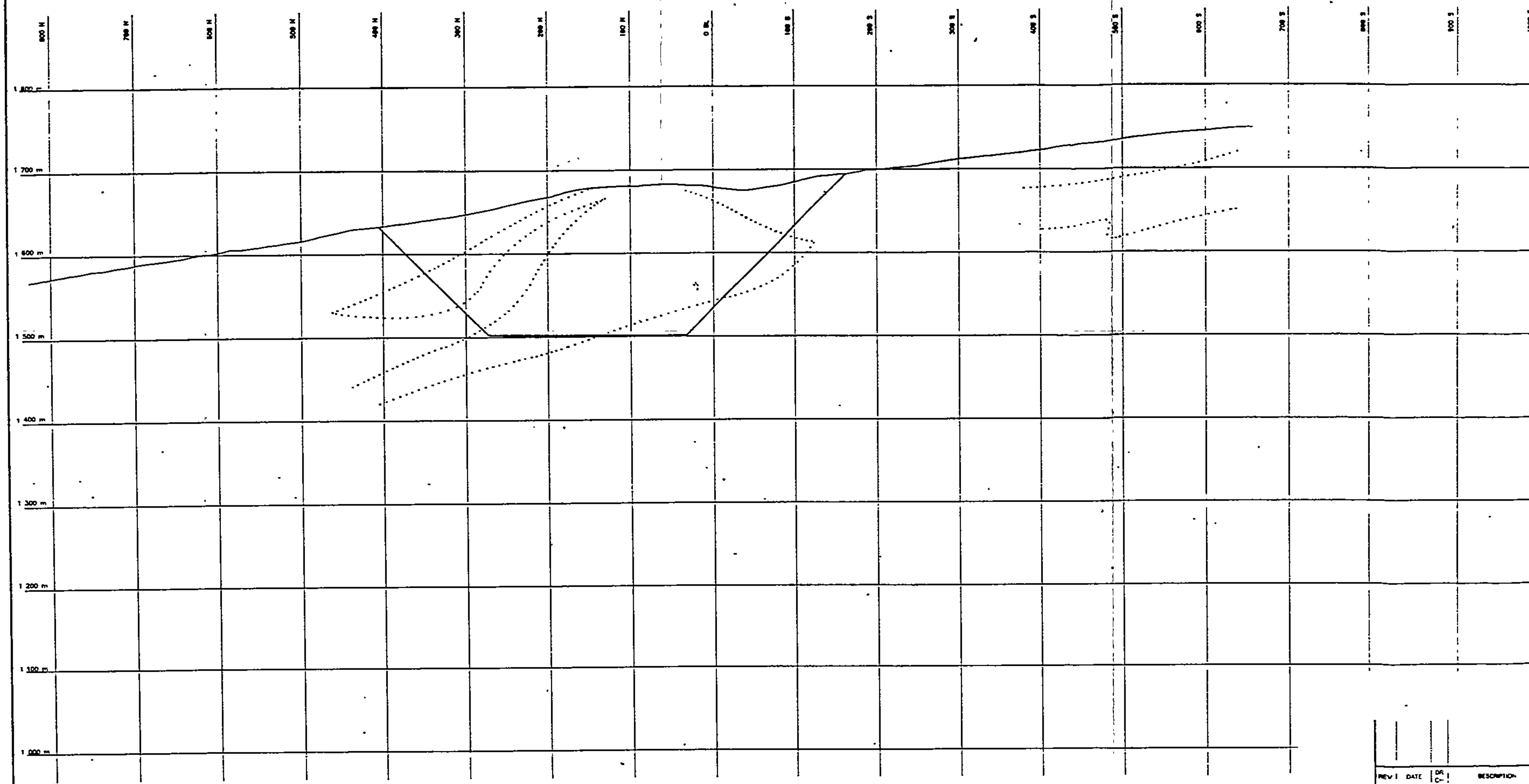
EAST ORE ZONE

SECTION 60W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.,J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV.
88005	HH017	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

EAST ORE ZONE

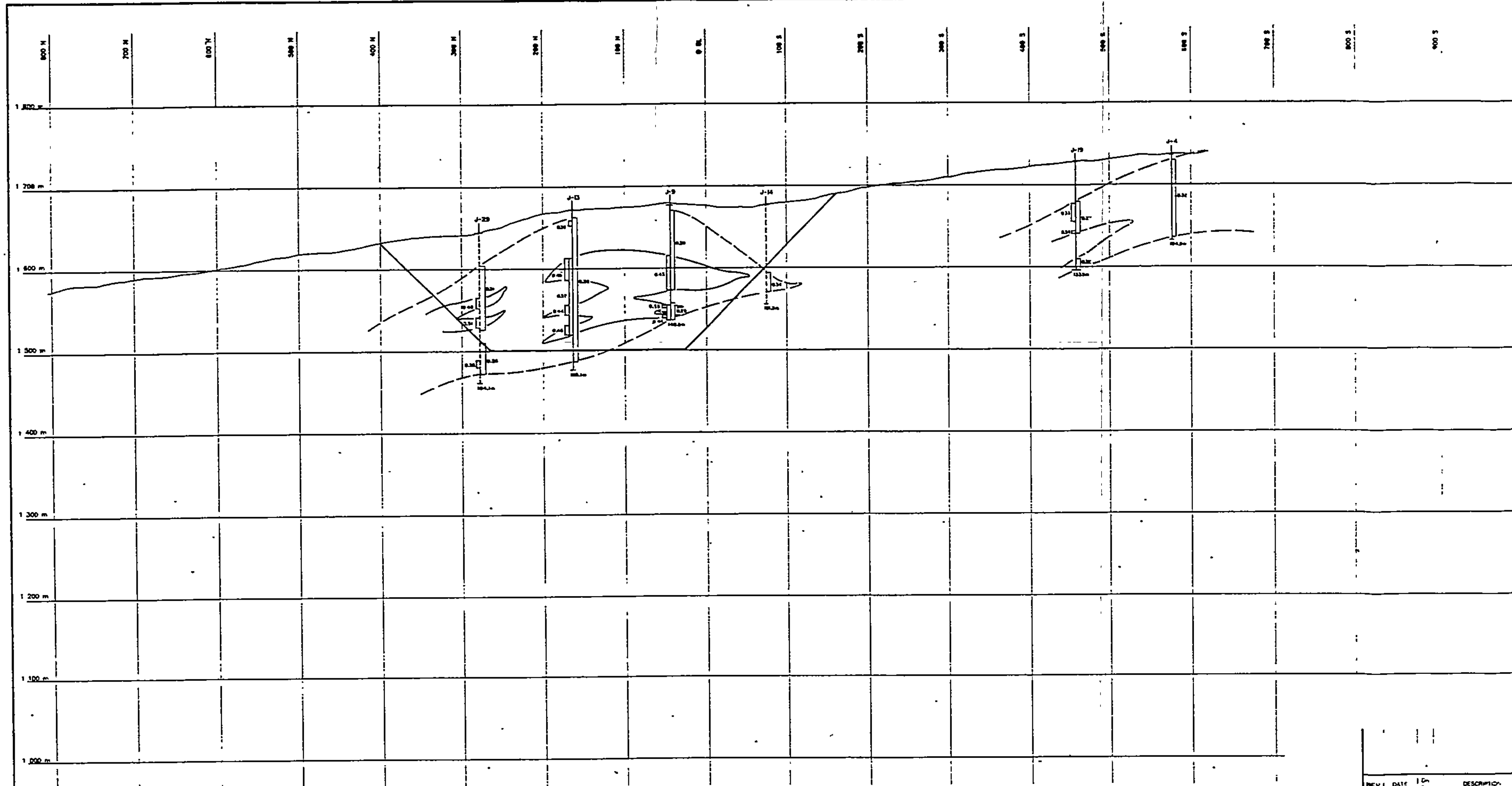
SECTION 120 W

DESIGNED	J.J.L.
DRAWN	V.J.A.J.R.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

SEAL

PROJECT No	DRAWING No	REV
88005	HH01B	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV | DATE | DESCRIPTION

Phillips Barrett Kaiser
Engineering Ltd

AURUN MINES LTD.

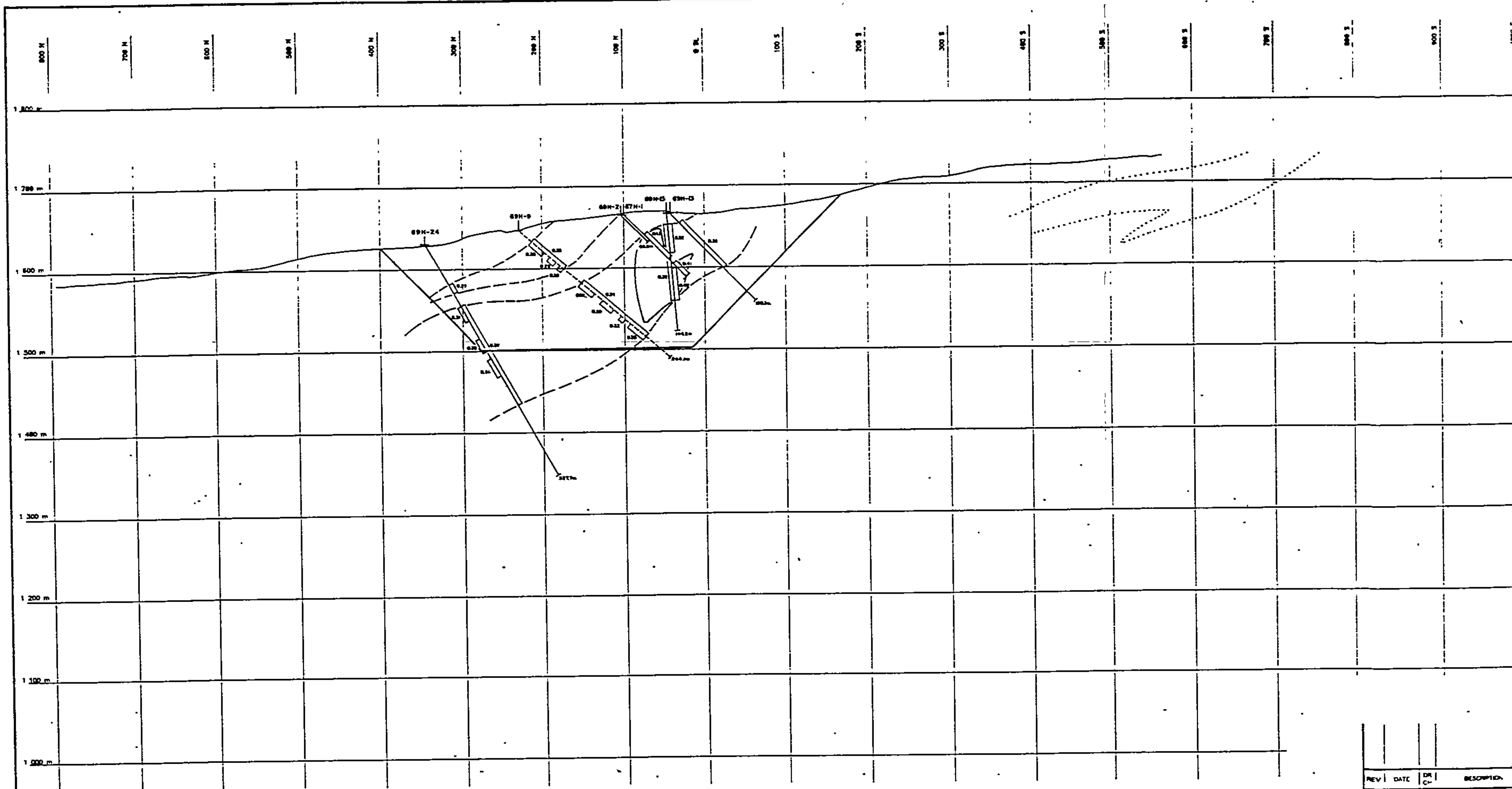
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 180 W

DESIGNED	J.J.L.
DRAWN	IV.J.A.J.R.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1:2000
DATE	88/03/30

PROJECT NO.	DRAWING NO.
88005	HH019

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Philips Barrett Kaiser
Engineering Ltd

AURUN MINES LTD.

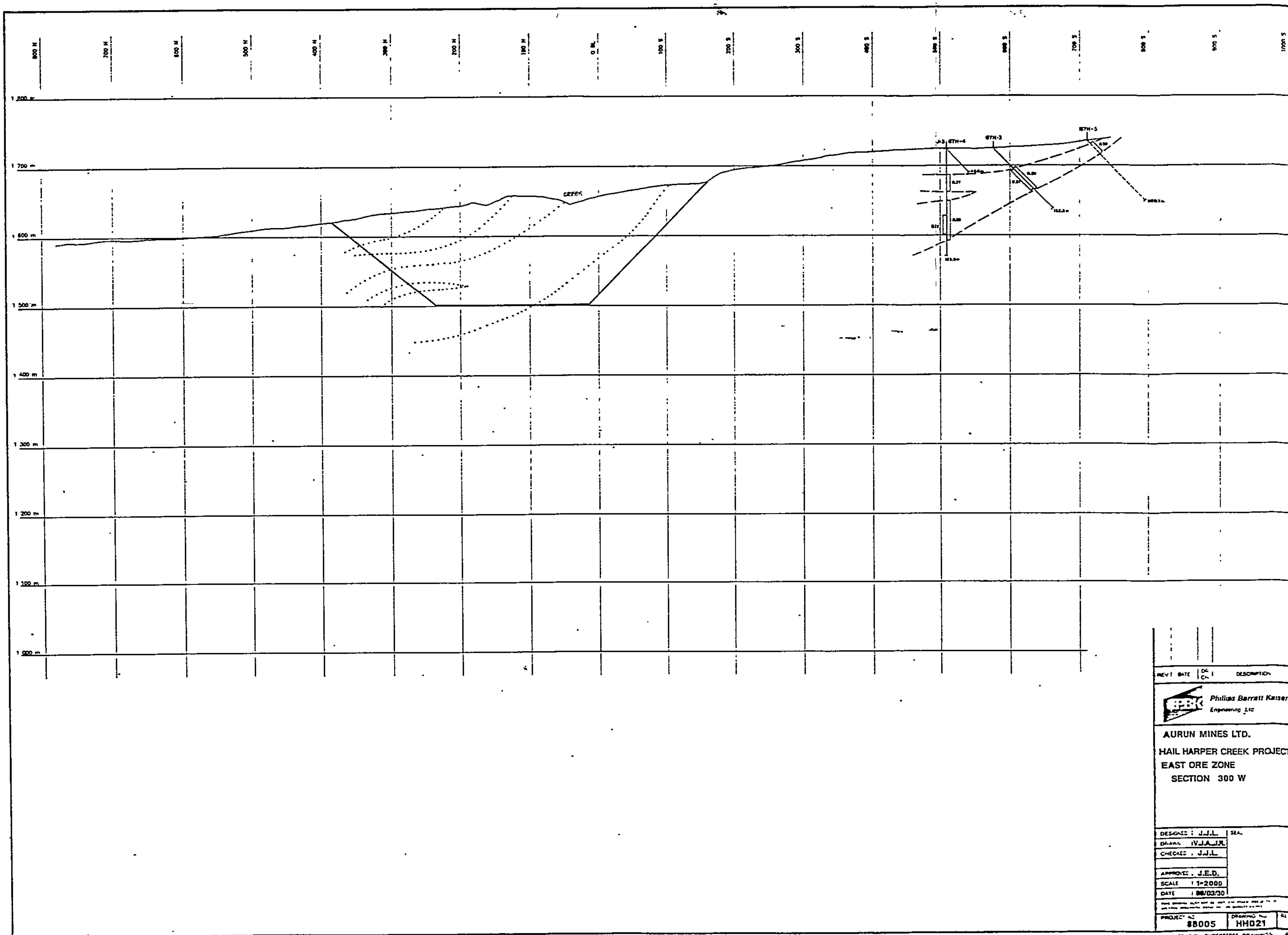
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 240 W

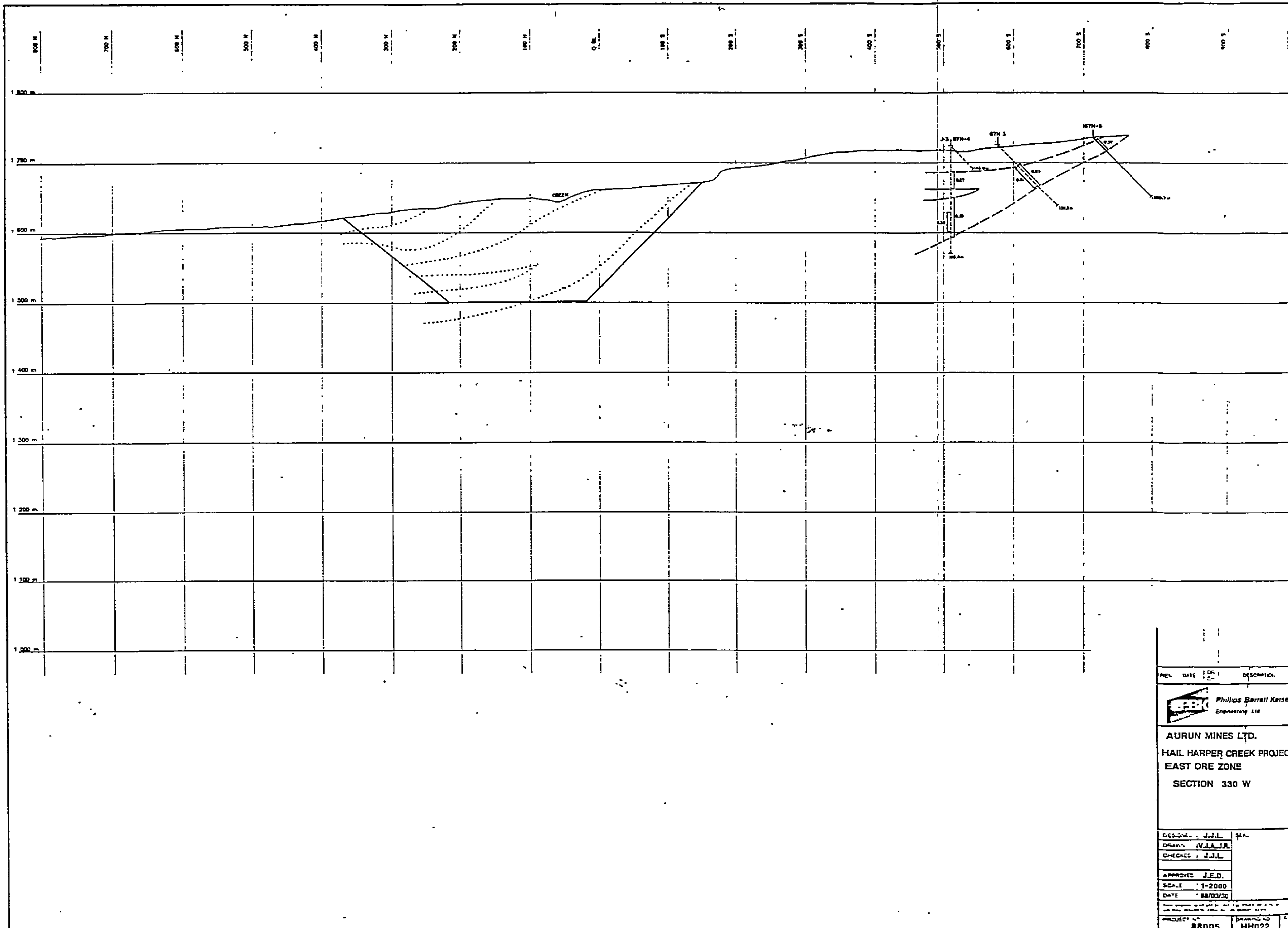
DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV.
88005	HH020	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



800 N
700 N
600 N
500 N
400 N
300 N
200 N
100 N
0 BL
100 S
200 S
300 S
400 S
500 S
600 S
700 S
800 S
900 S
1000 S
1 800 m
1 700 m
1 600 m
1 500 m
1 400 m
1 300 m
1 200 m
1 100 m
1 000 m
CREEK
87H-4
87H-3
87H-5
REV DATE DR CH DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SECTION 300 W
DESIGNED : J.J.L.
SEAL
DRAWN : V.J.A.,J.R.
CHECKED : J.J.L.
APPROVED : J.E.D.
SCALE : 1=2000
DATE : 88/03/30
PROJECT No. 88005
DRAWING No. HH021
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



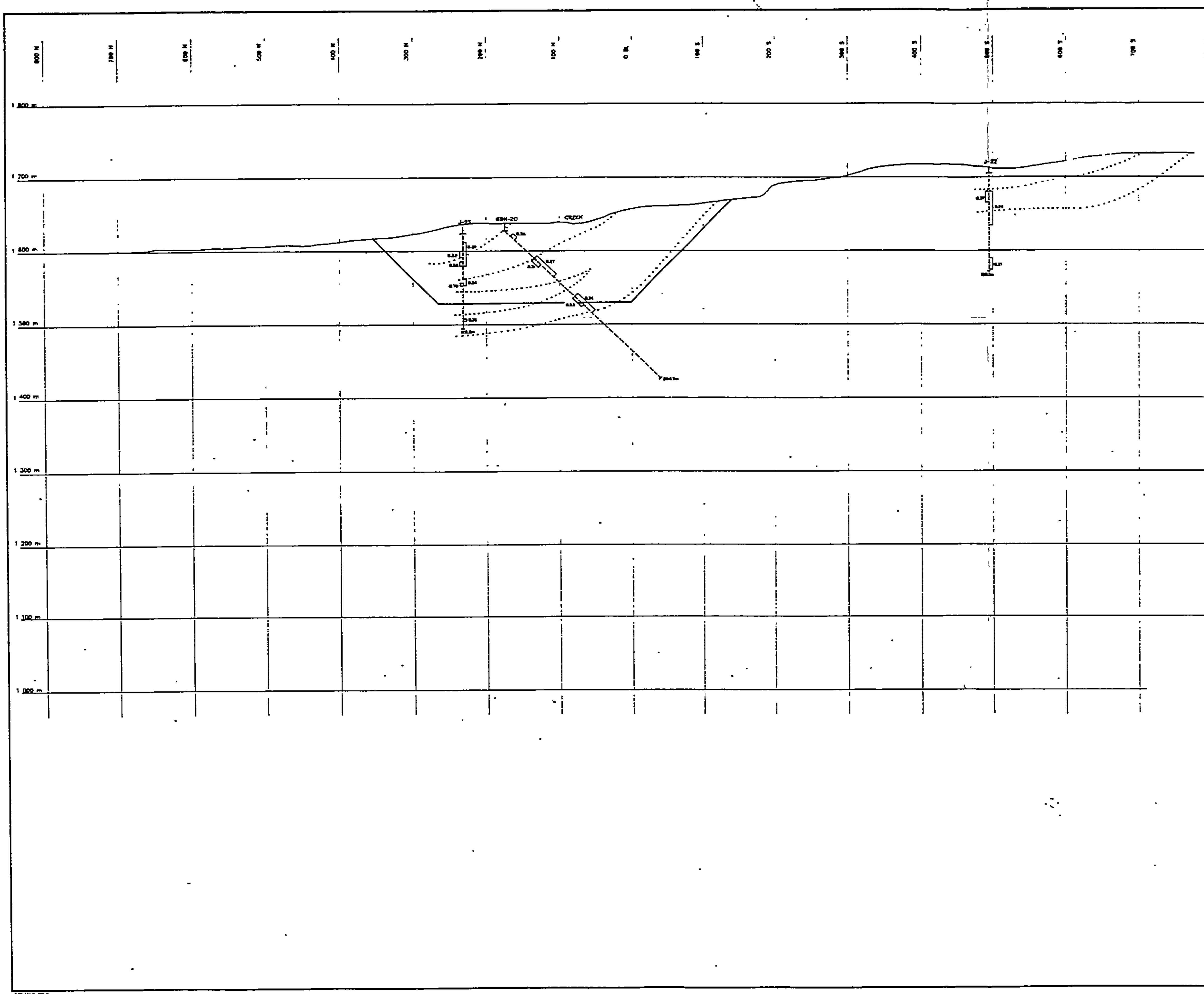
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
EAST ORE ZONE
SECTION 330 W

Phillips Barratt Kaiser Engineering Ltd

DESIGNED	J.J.L.
DRAWN	V.J.A.J.R.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

PROJECT No. 88005 DRAWING No. HH022

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

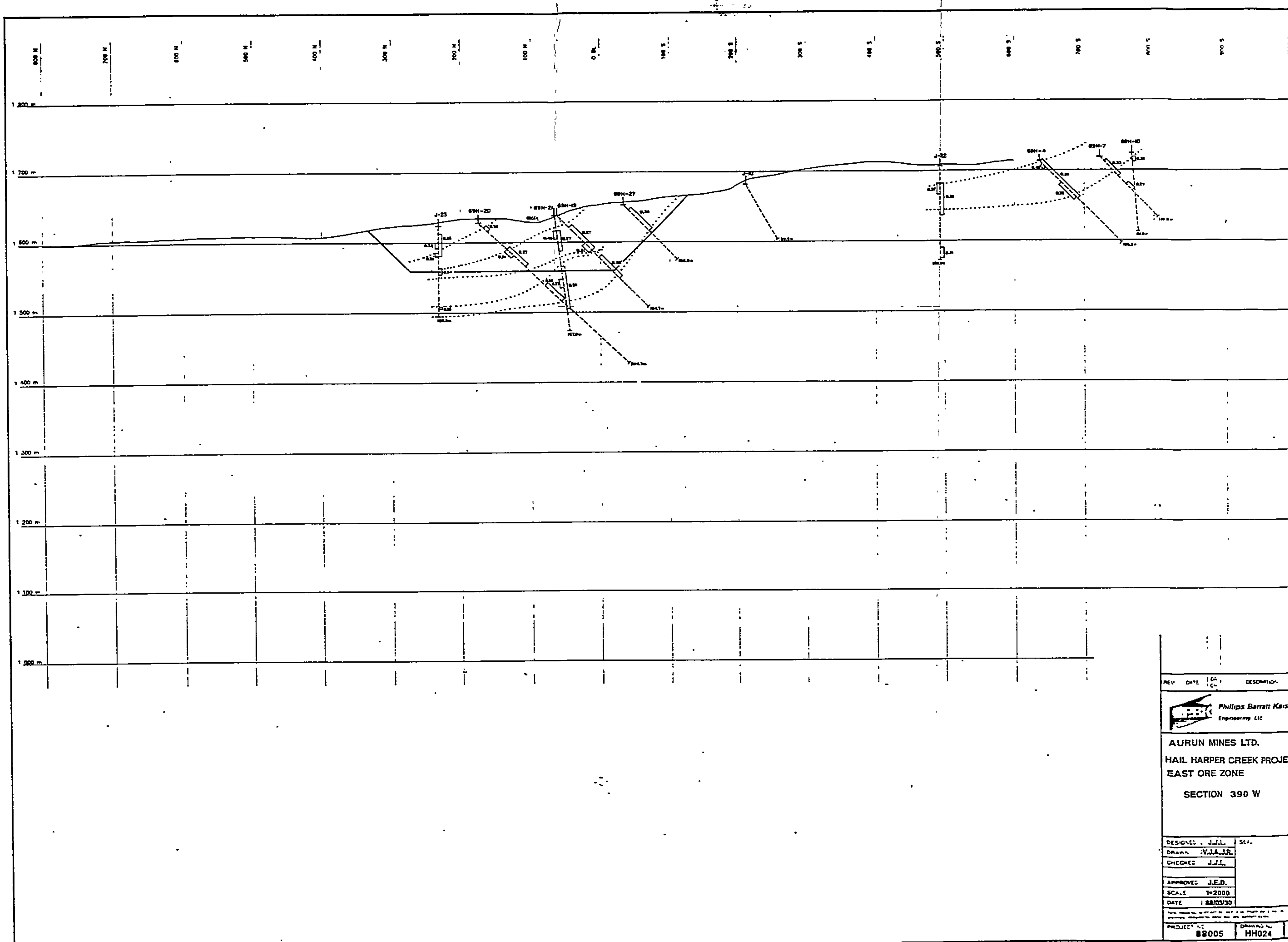
EAST ORE ZONE

SECTION 360 W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH023	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

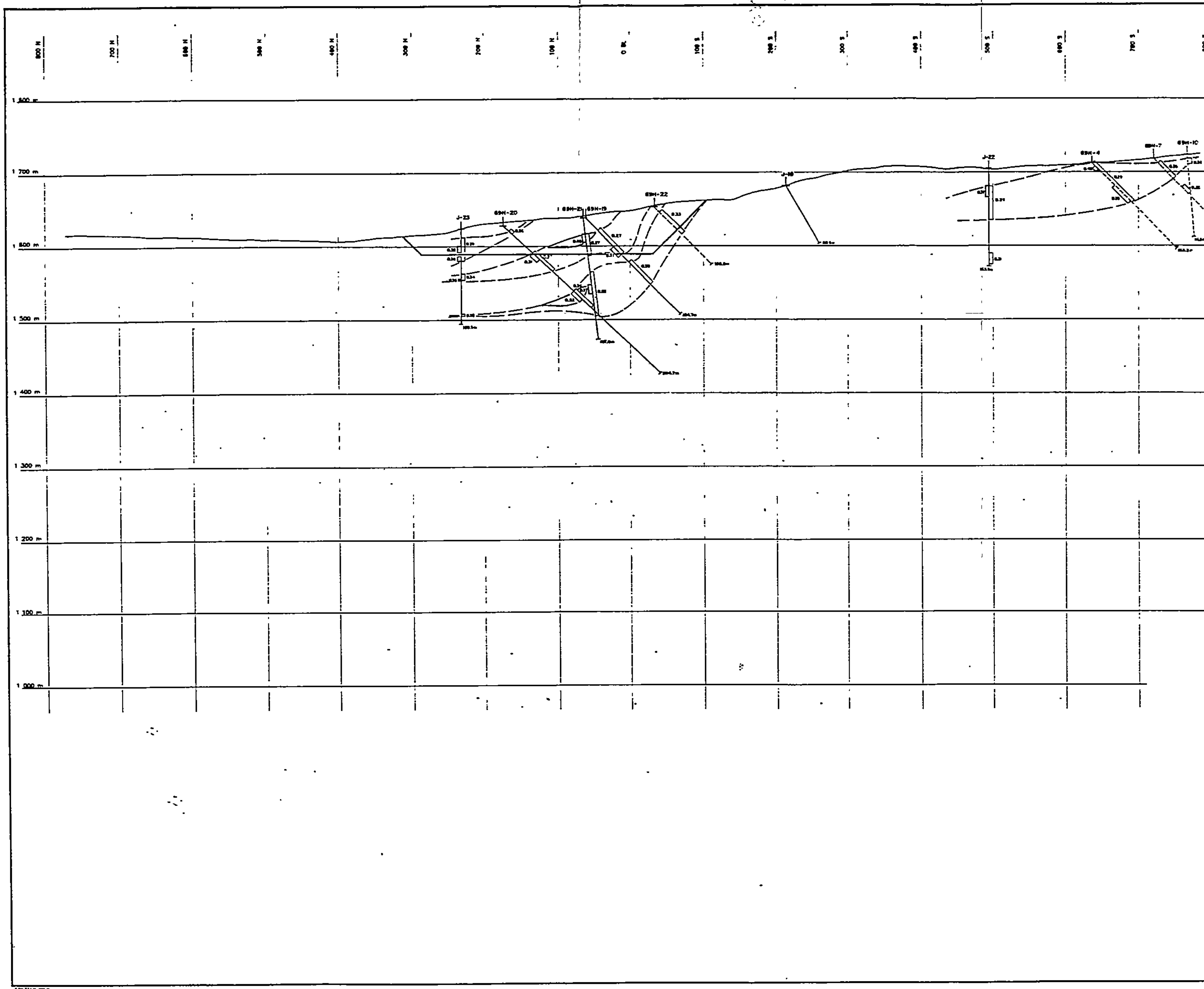
HAIL HARPER CREEK PROJECT
EAST ORE ZONE

SECTION 390 W

DESIGNED	J.J.L.
DRAWN	V.J.A.J.R.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

PROJECT No	DRAWING No
88005	HH024

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barrett Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

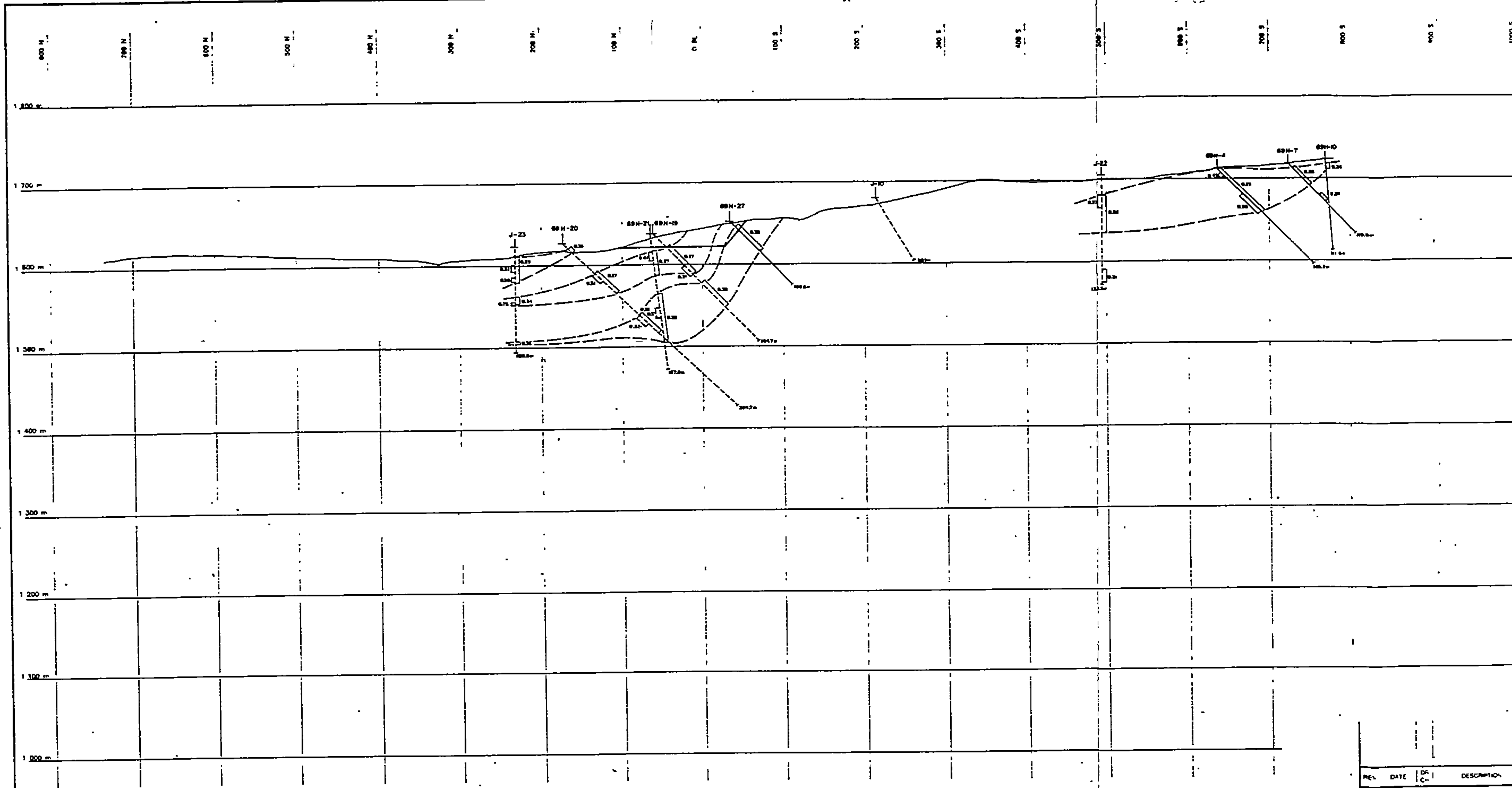
EAST ORE ZONE

SECTION 420 W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A./J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No.	DRAWING NO.	REV.
88005	HH025	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR. / CH.	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd.

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

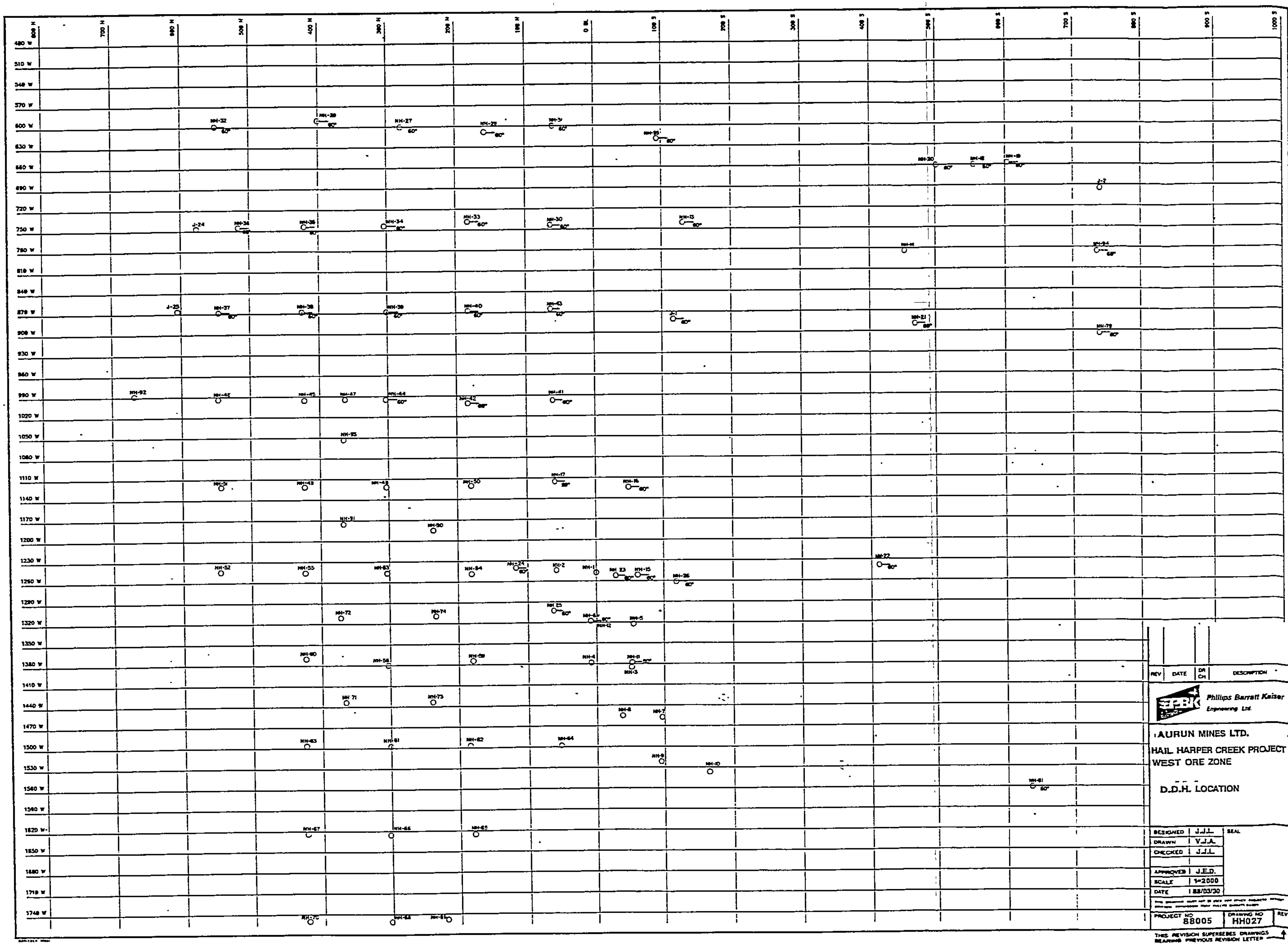
EAST ORE ZONE

SECTION 450 W

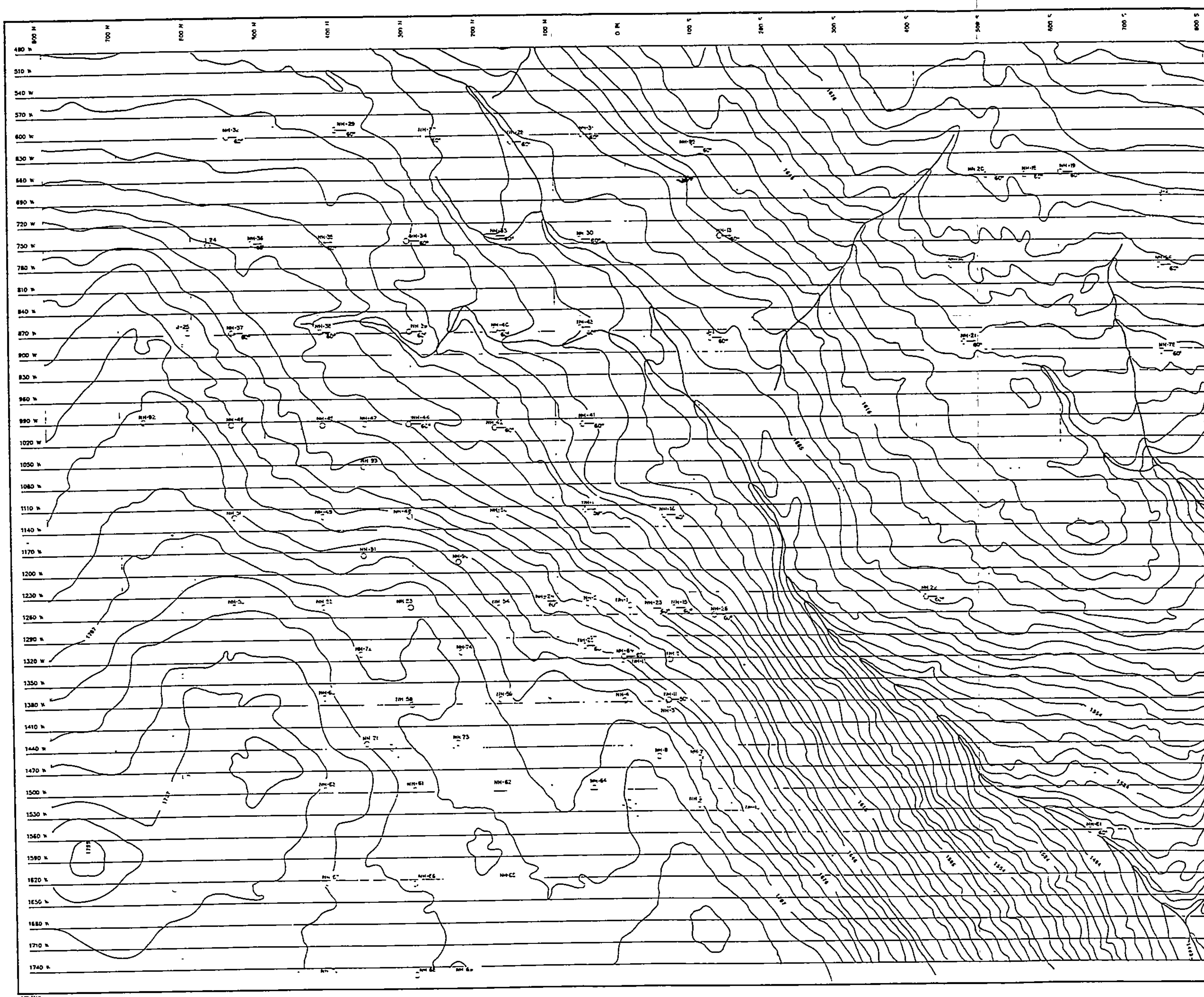
DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.J.R.	
CHECKED	J.J.L.	
APPROVED	J.E.D	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No.	DRAWING No.	REV.
88005	HH026	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV
DATE
DR CH
DESCRIPTION
Phillips Barratt Kaiser
Engineering Ltd.
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE
D.D.H. LOCATION
DESIGNED J.J.L.
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1=2000
DATE 88/03/30
SEAL
PROJECT NO 88005
DRAWING NO HH027
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

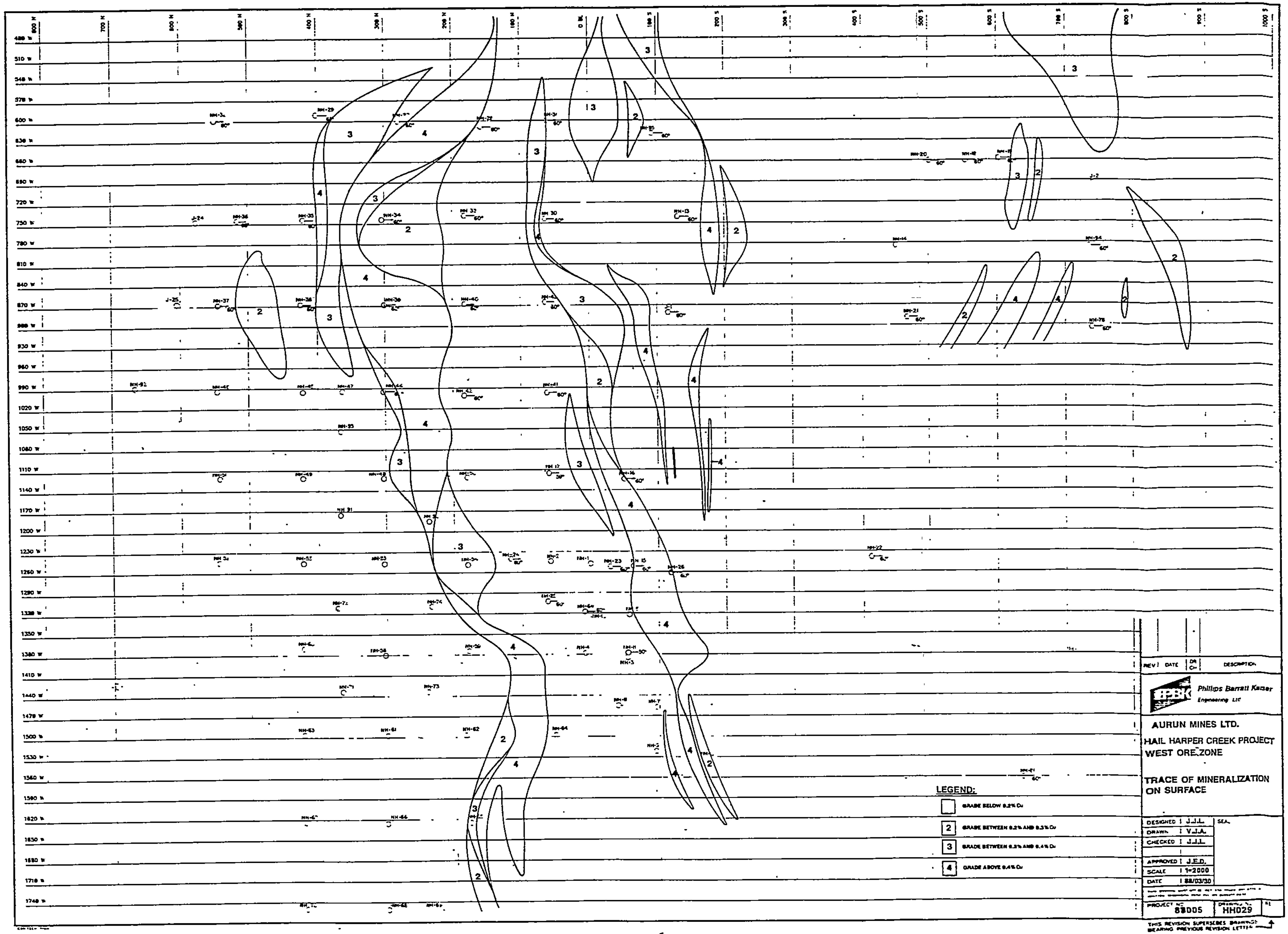
HAIL HARPER CREEK PROJECT
WEST ORE ZONE

SURFACE PLAN

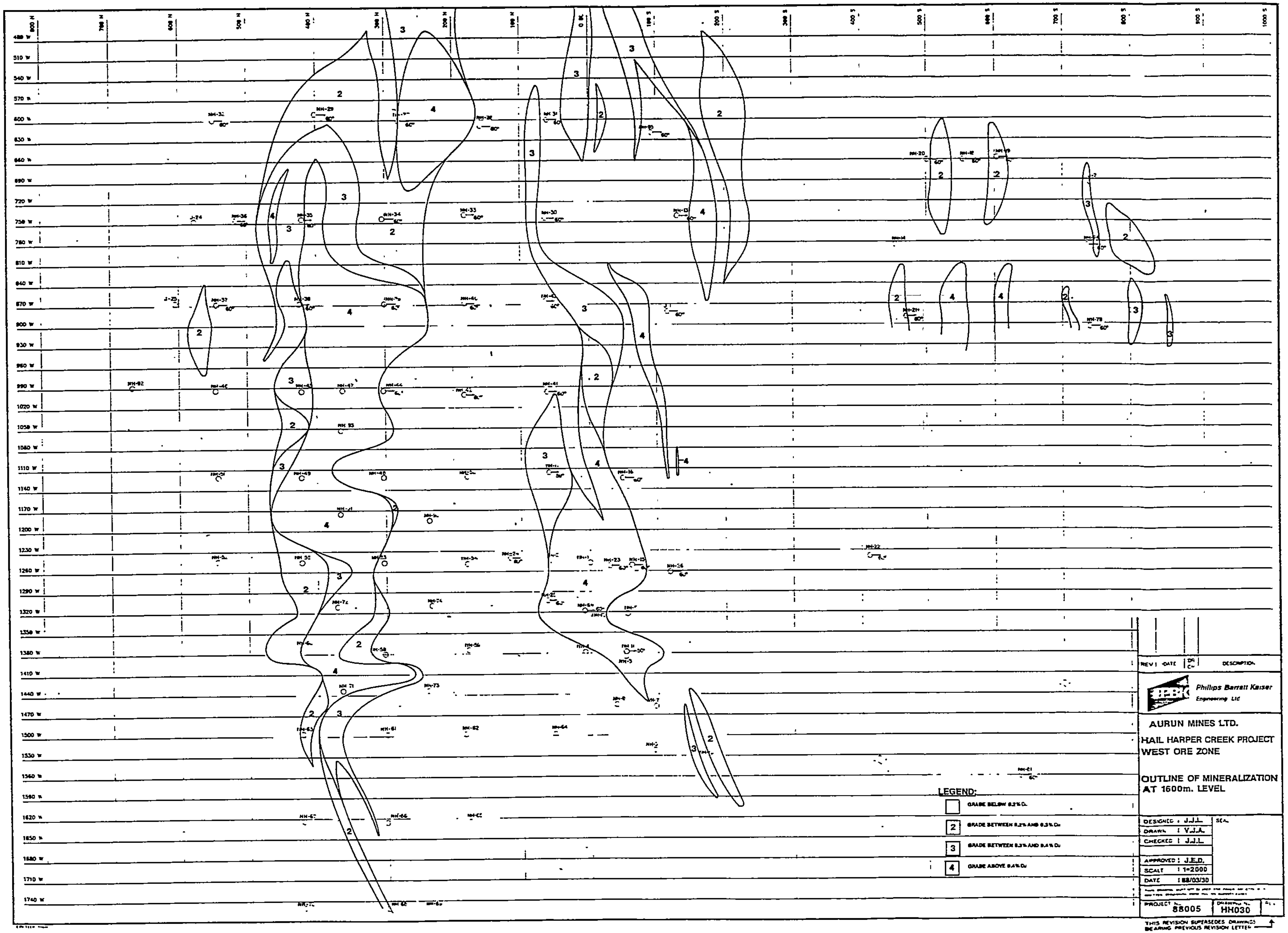
DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT No.	DRAWING No.	REV.
88005	HH028	

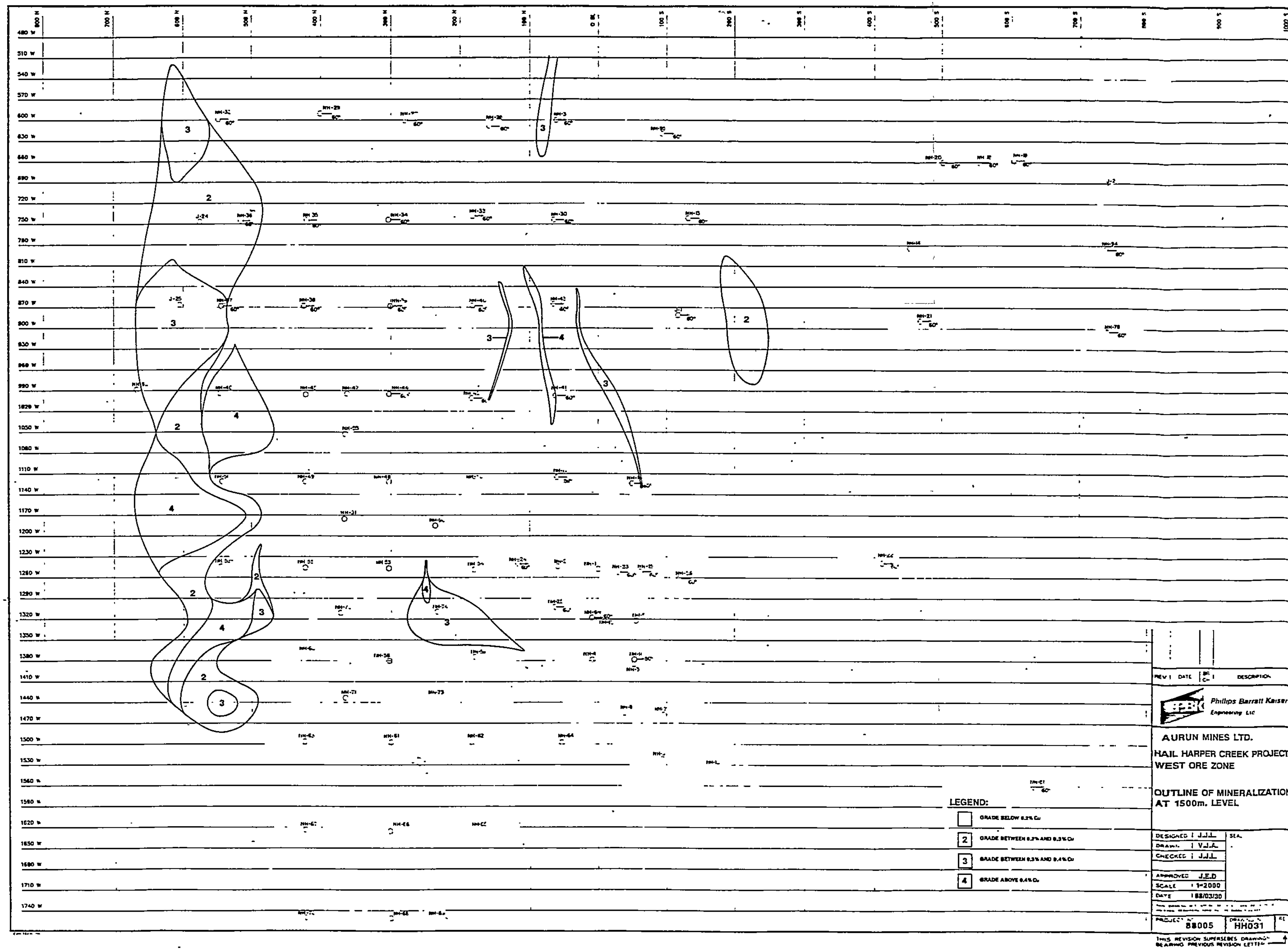
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER

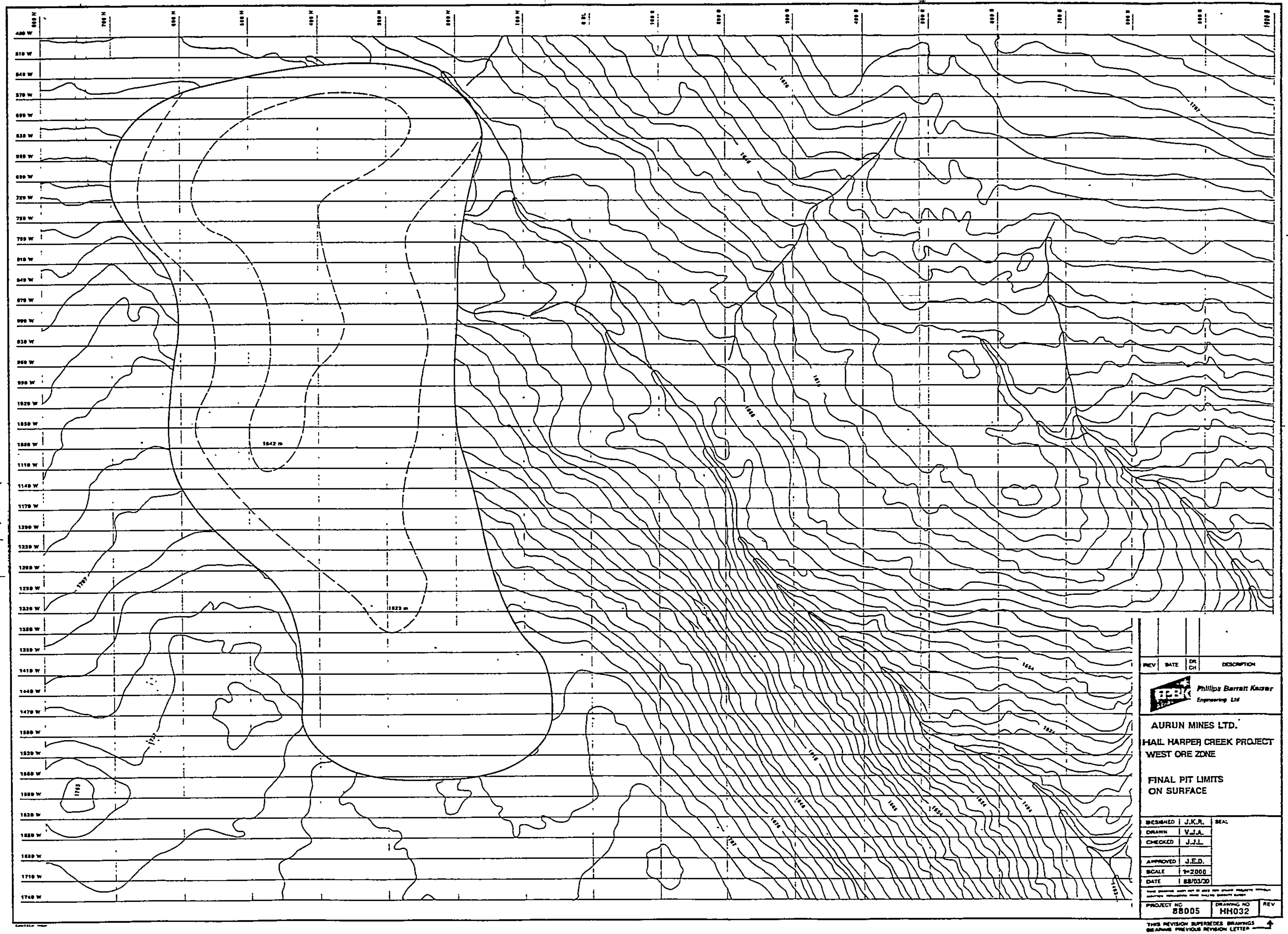


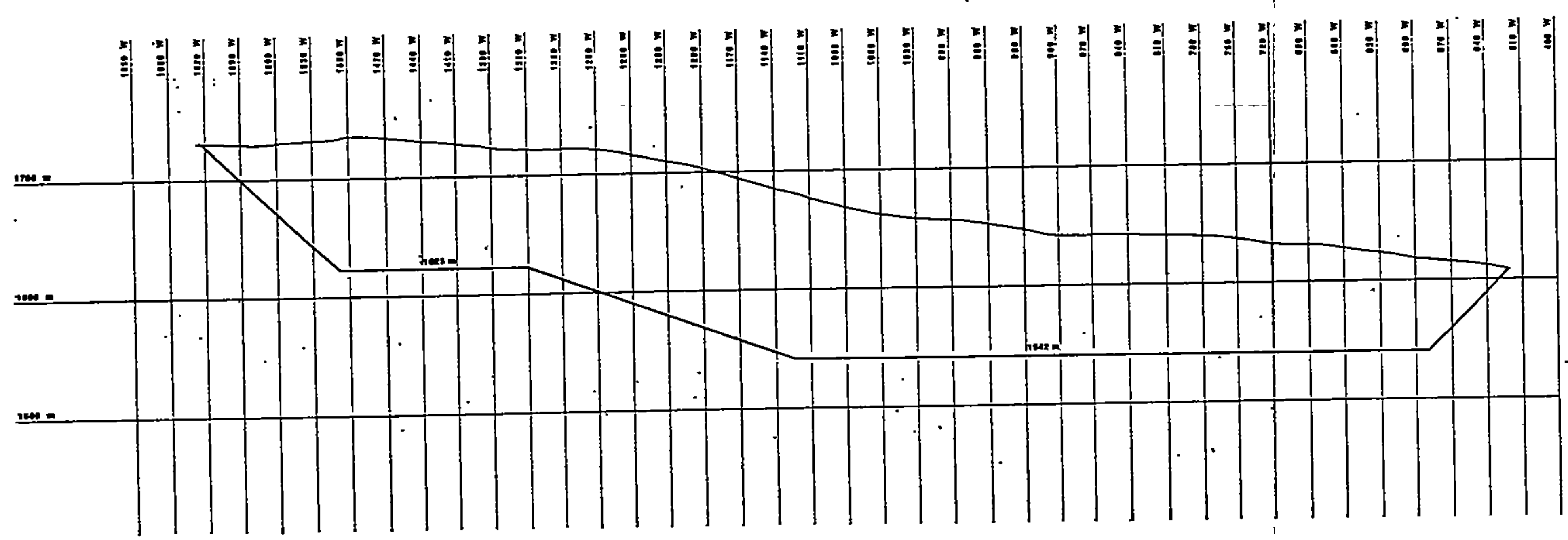
LEGEND:
GRADE BELOW 0.2% Cu
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV DATE DR CH DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE
TRACE OF MINERALIZATION ON SURFACE
DESIGNED J.J.L.
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1=2000
DATE 88/03/30
SEAL
PROJECT NO 88005
DRAWING No HH029
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



LEGEND:
GRADE BELOW 0.2% Cu
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV | DATE | DR CH | DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE
OUTLINE OF MINERALIZATION
AT 1600m. LEVEL
DESIGNED : J.J.L.
DRAWN : V.J.A.
CHECKED : J.J.L.
APPROVED : J.E.D.
SCALE : 1=2000
DATE : 88/03/30
SEAL
PROJECT No. 88005
DRAWING No. HH030
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


LEGEND:
GRADE BELOW 0.2% Cu
2 GRADE BETWEEN 0.2% AND 0.3% Cu
3 GRADE BETWEEN 0.3% AND 0.4% Cu
4 GRADE ABOVE 0.4% Cu
REV | DATE | DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE
OUTLINE OF MINERALIZATION
AT 1500m. LEVEL
DESIGNED J.J.L.
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D
SCALE 1=2000
DATE 88/03/30
PROJECT Nº 88005
DRAWING Nº HH031
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


REV
DATE
DR CH
DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE
FINAL PIT LIMITS
ON SURFACE
DESIGNED J.K.R.
SEAL
DRAWN V.J.A.
CHECKED J.J.L.
APPROVED J.E.D.
SCALE 1=2000
DATE 88/03/30
PROJECT NO 88005
DRAWING NO HH032
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER
1642 m
1623 m



REV.	DATE	DR. CH.	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd.

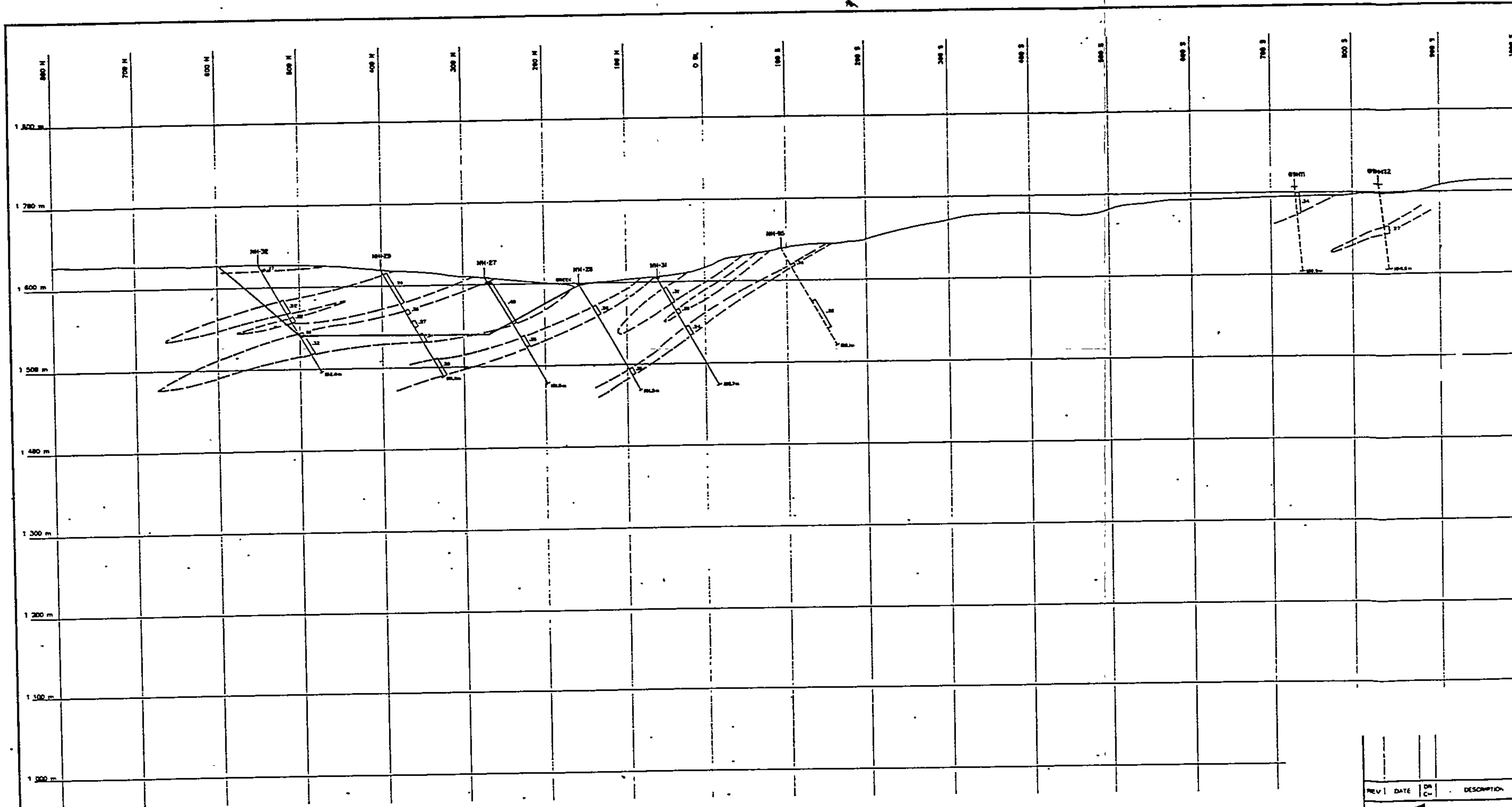
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
WEST ORE ZONE

FINAL PIT BOTTOM PROFILE

DESIGNED	J.K.R.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH033	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd.

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

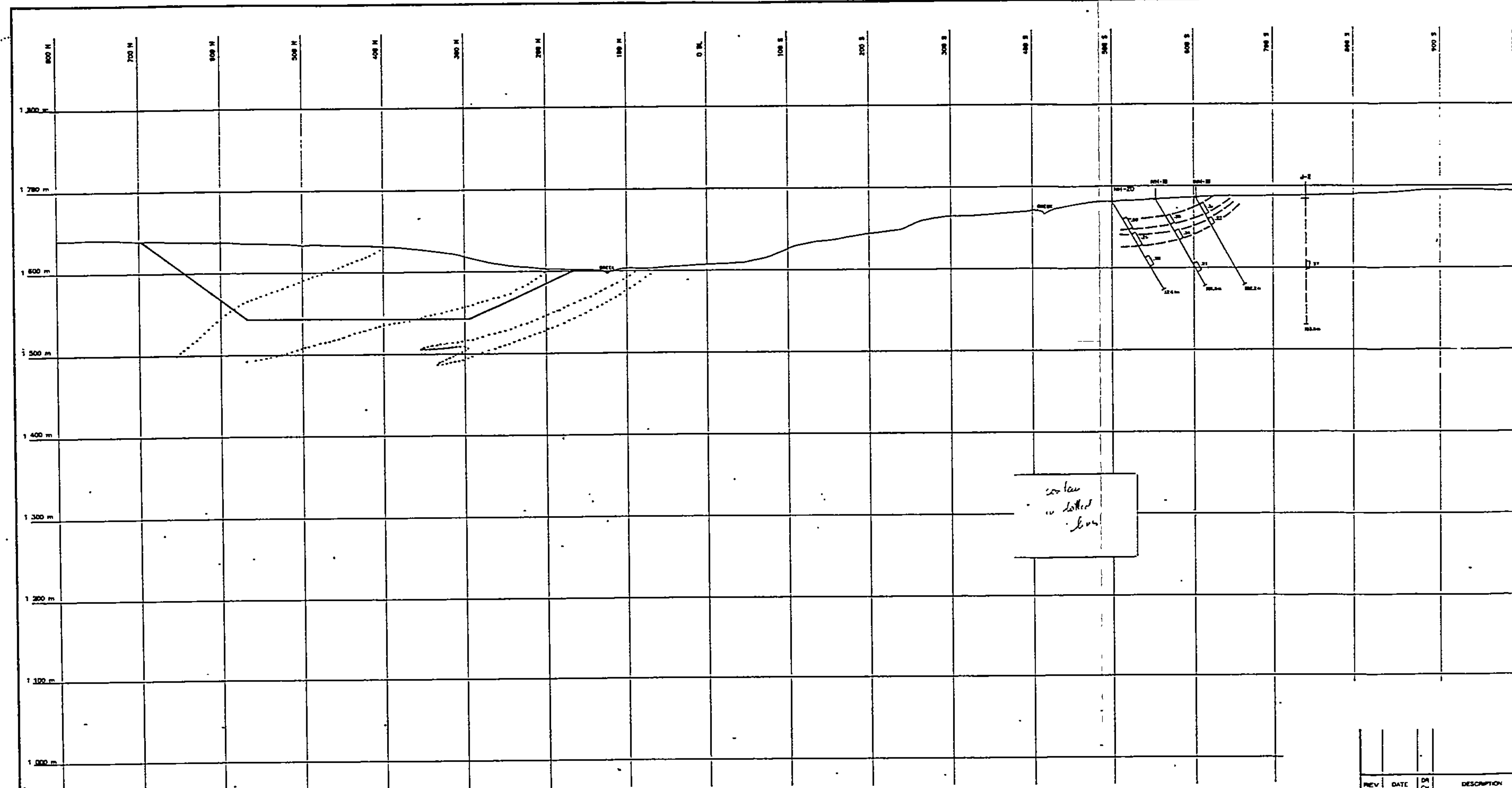
SECTION 600W

DESIGNED	J.J.L.
DRAWN	V.J.A.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

SEAL

PROJECT No	DRAWING No	REV
88005	HH034	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

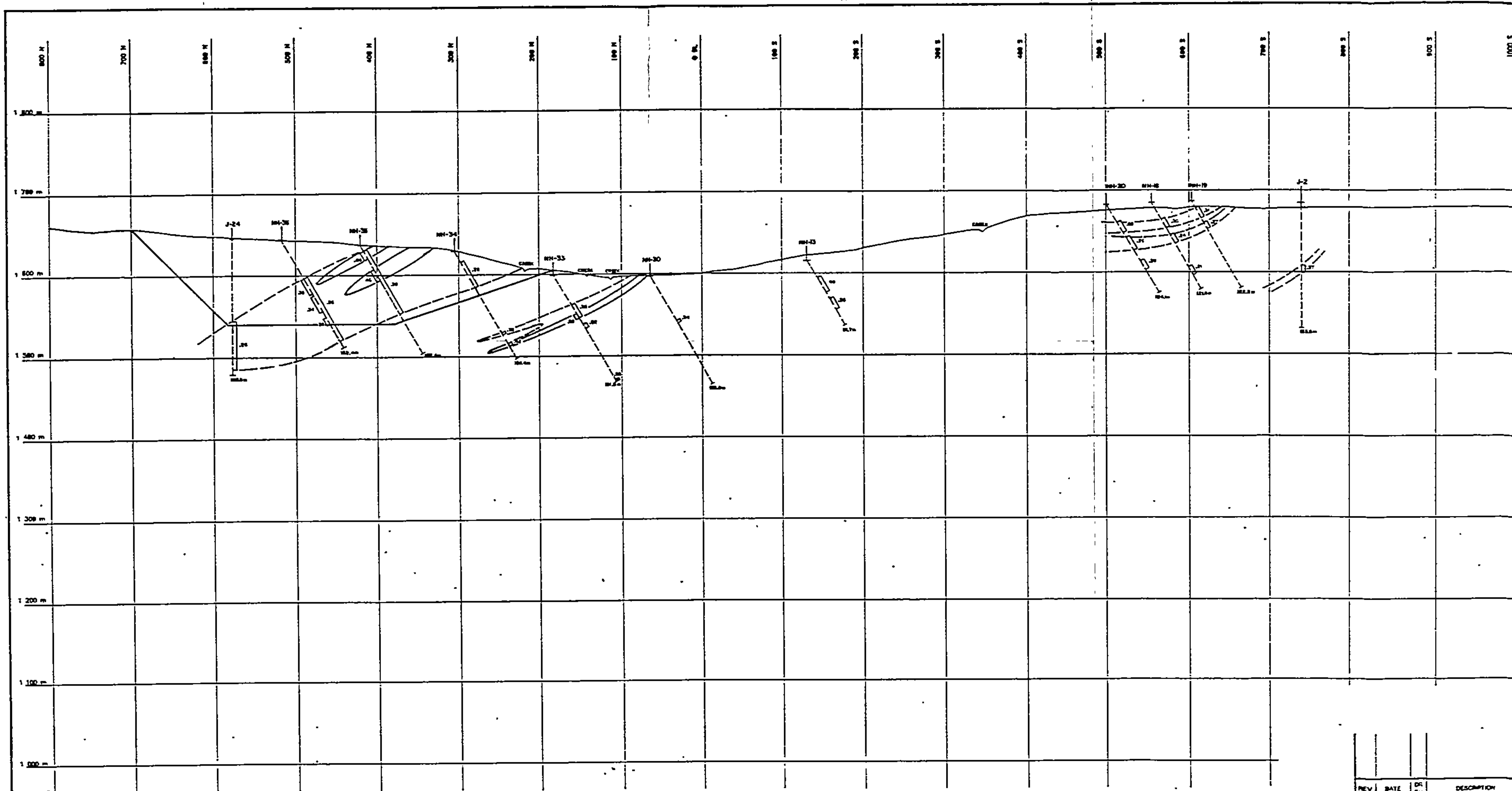
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 660W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH035	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

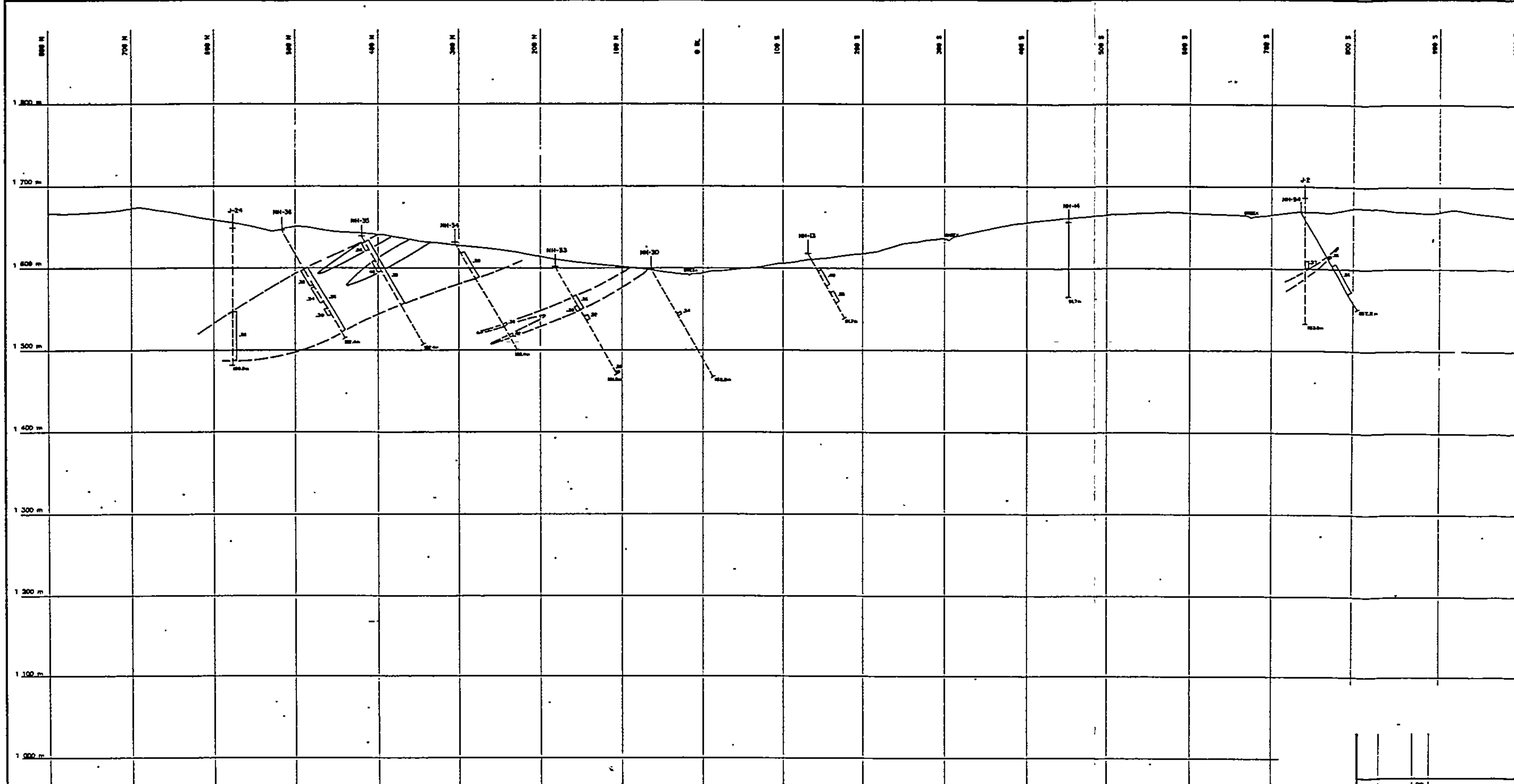
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 720W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH036	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barrett Kaiser Engineering Ltd

AURUN MINES LTD.

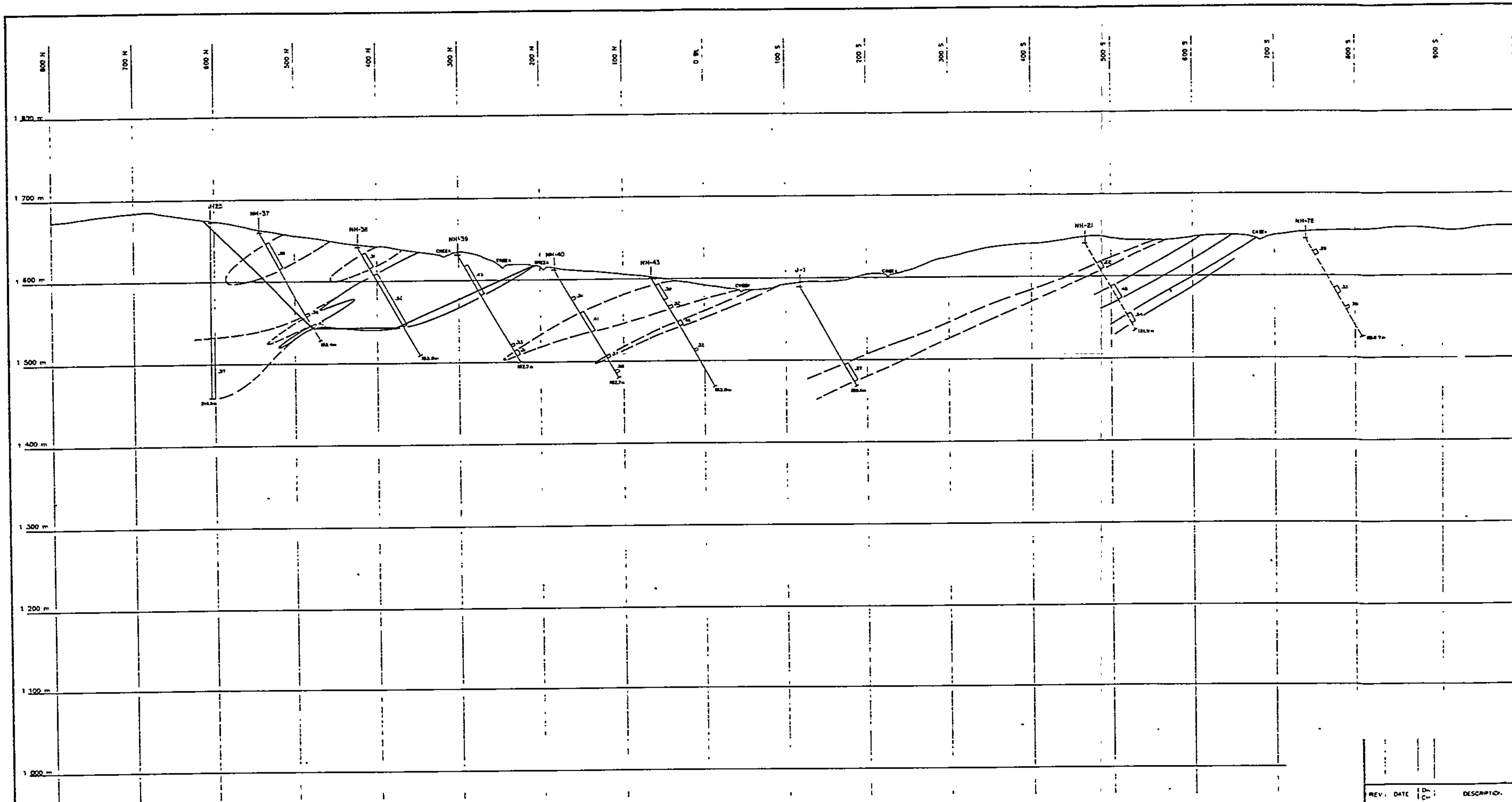
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 780W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV
88005	HH037	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR. / CH.	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

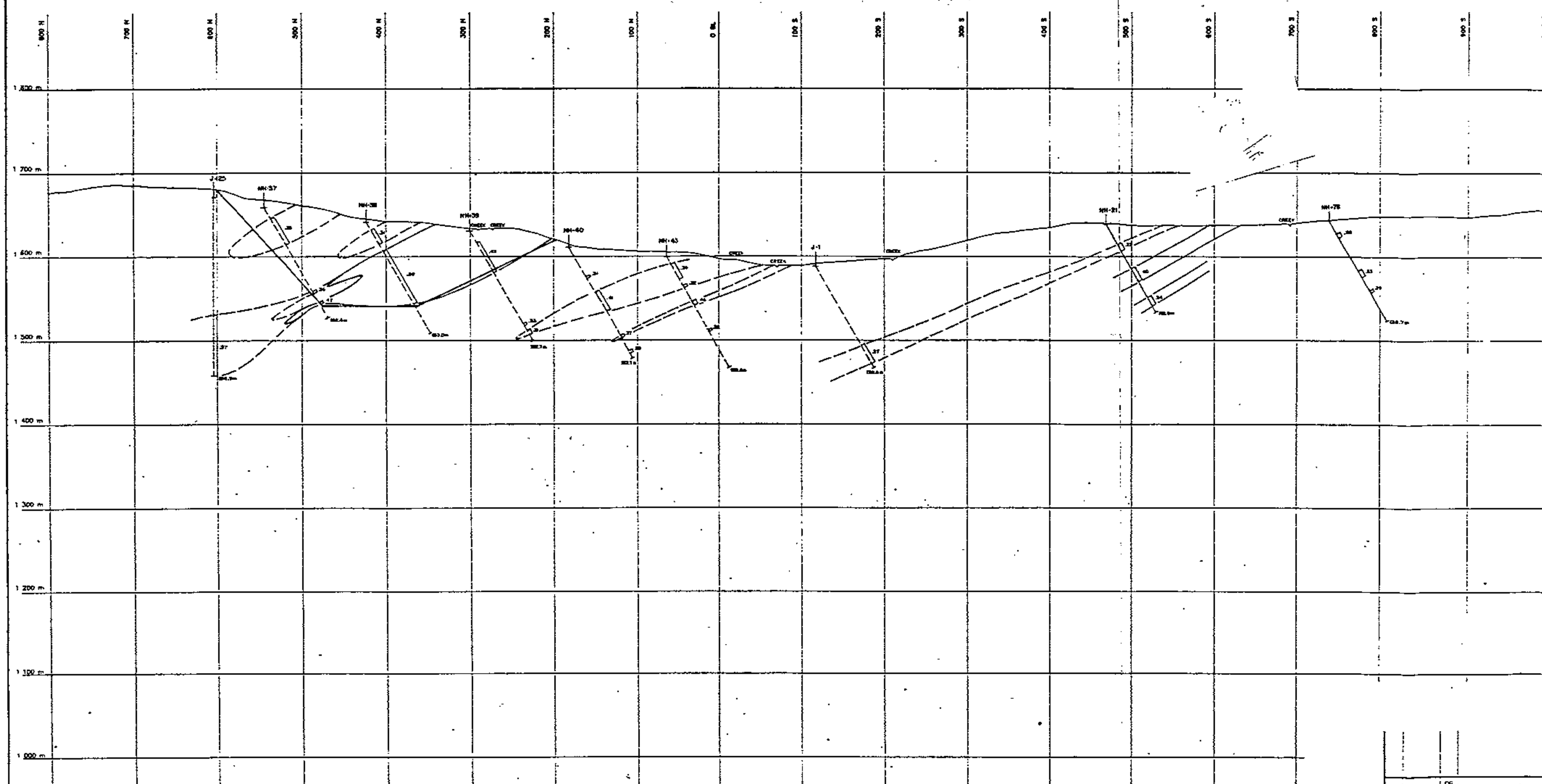
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 870W

DESIGNED	J.J.L.
DRAWN	V.J.A.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

PROJECT No.	DRAWING No.	REV.
88005	HH038	

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

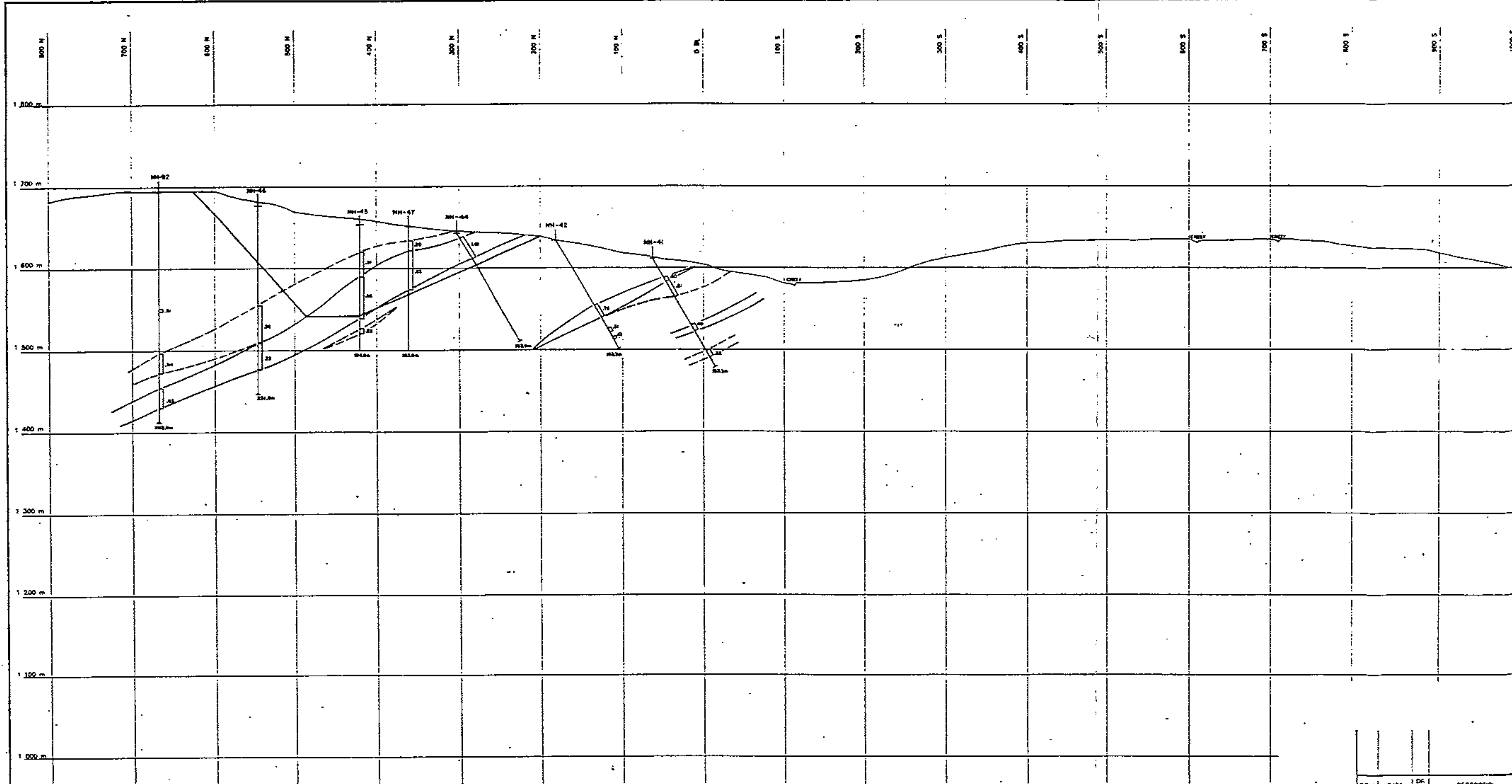
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 900W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH039	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR. CH.	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

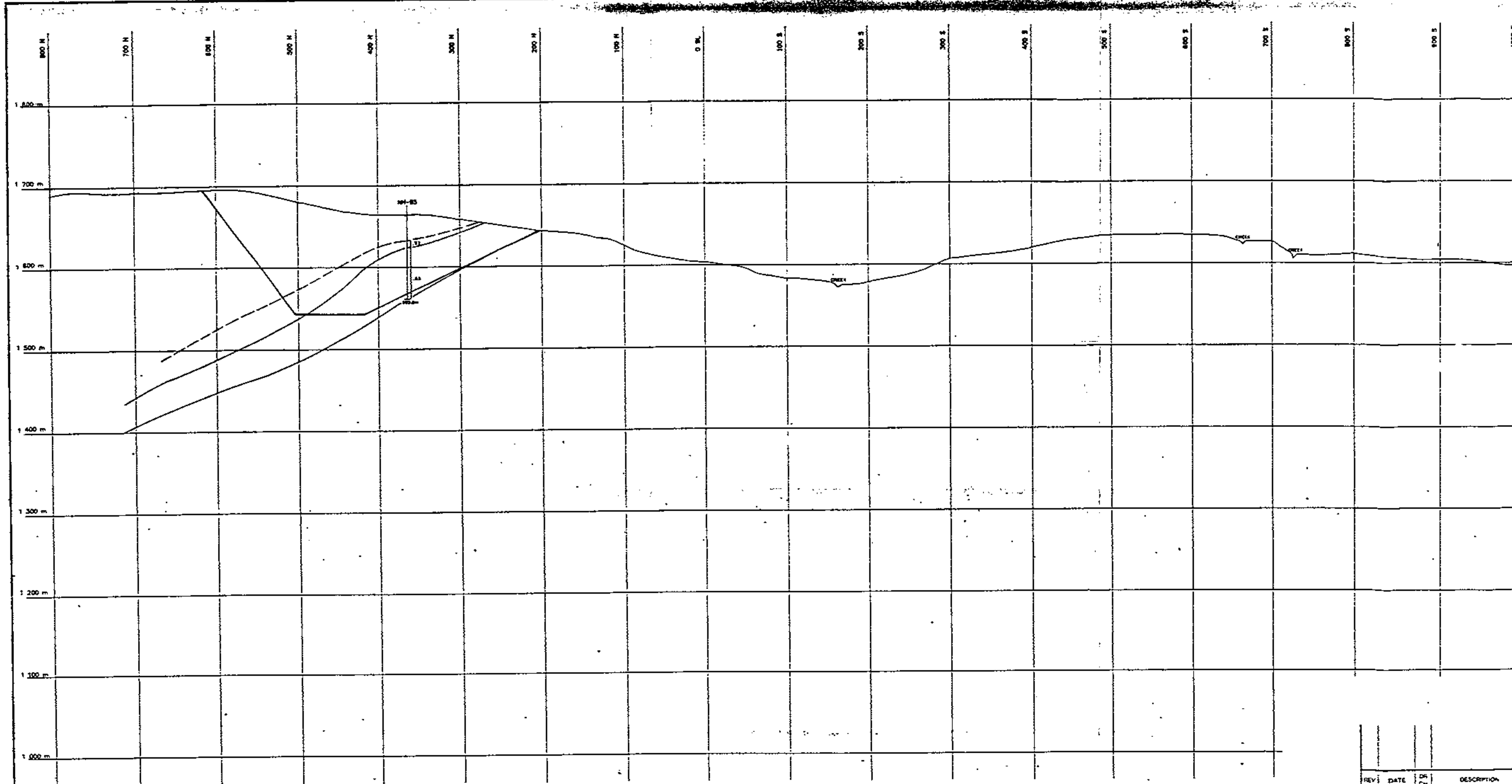
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL
PIT PROFILE

SECTION 990W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV
88005	HH040	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

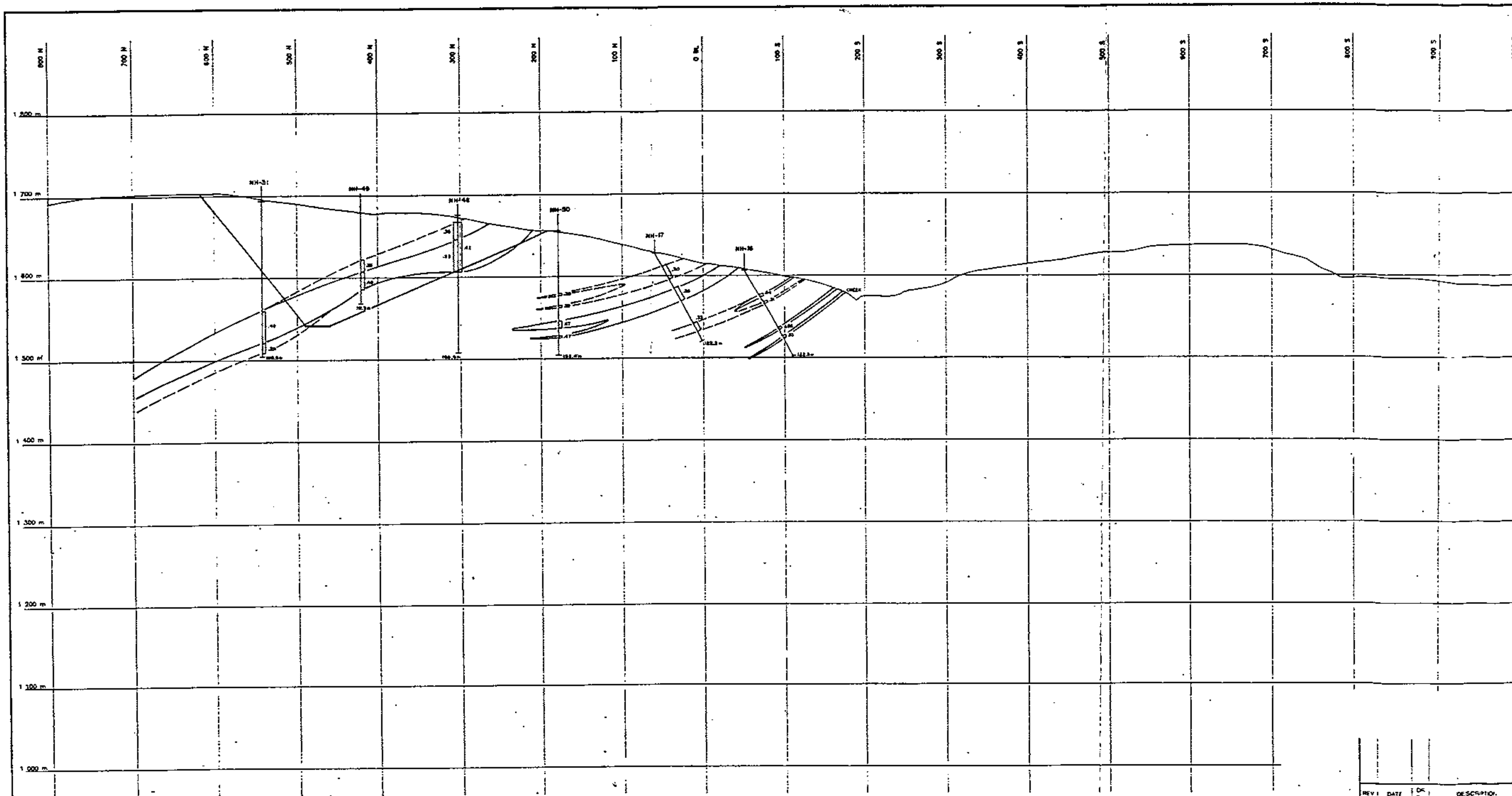
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1050W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH041	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips-Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

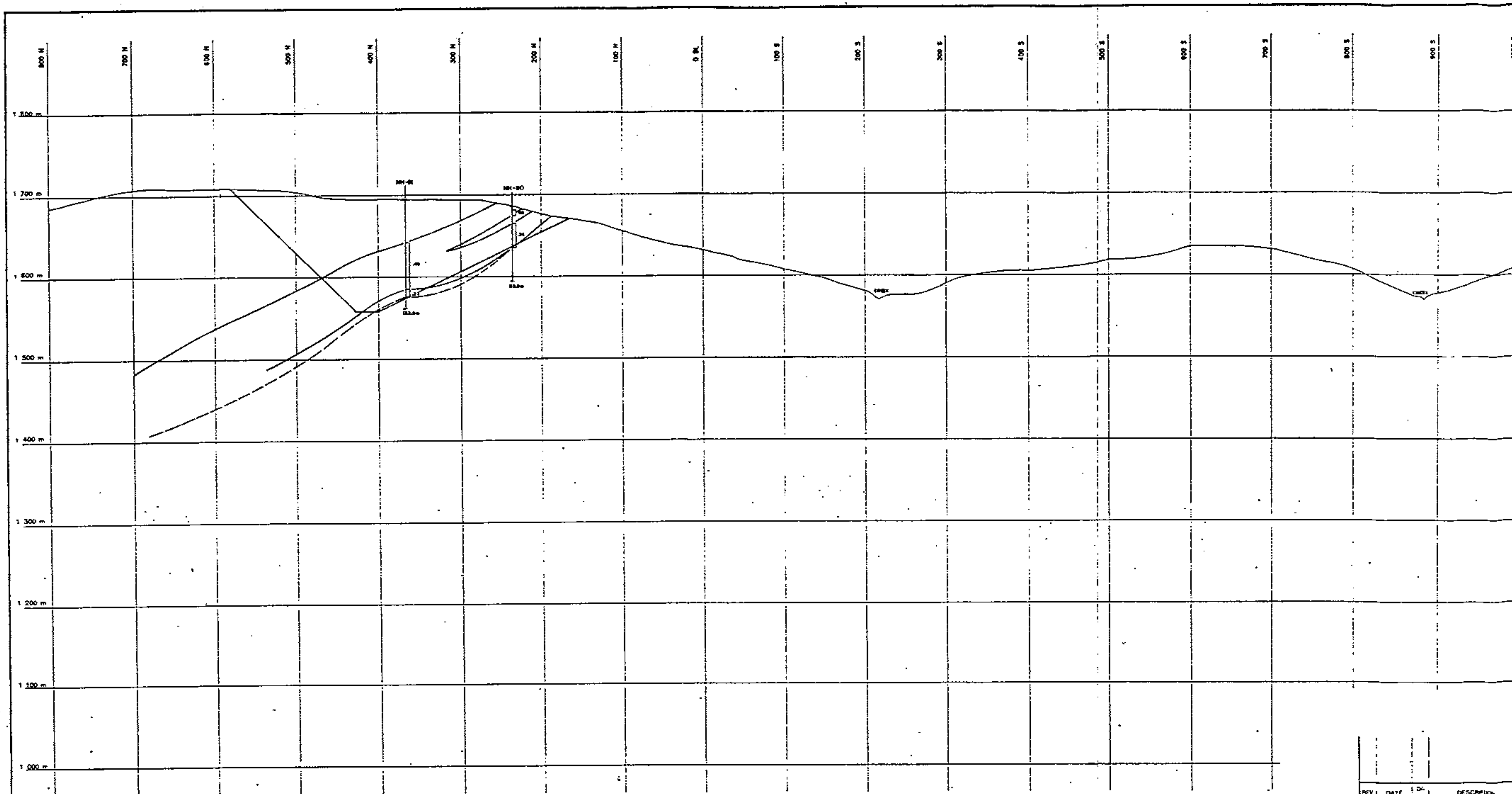
SECTION 1110W

DESIGNED	J.J.L.
DRAWN	V.J.A.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

SEAL

PROJECT NO.	DRAWING NO.	REV.
88005	HH042	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR / CH	DESCRIPTION

Phillips Barrett Kaiser Engineering Ltd

AURUN MINES LTD.

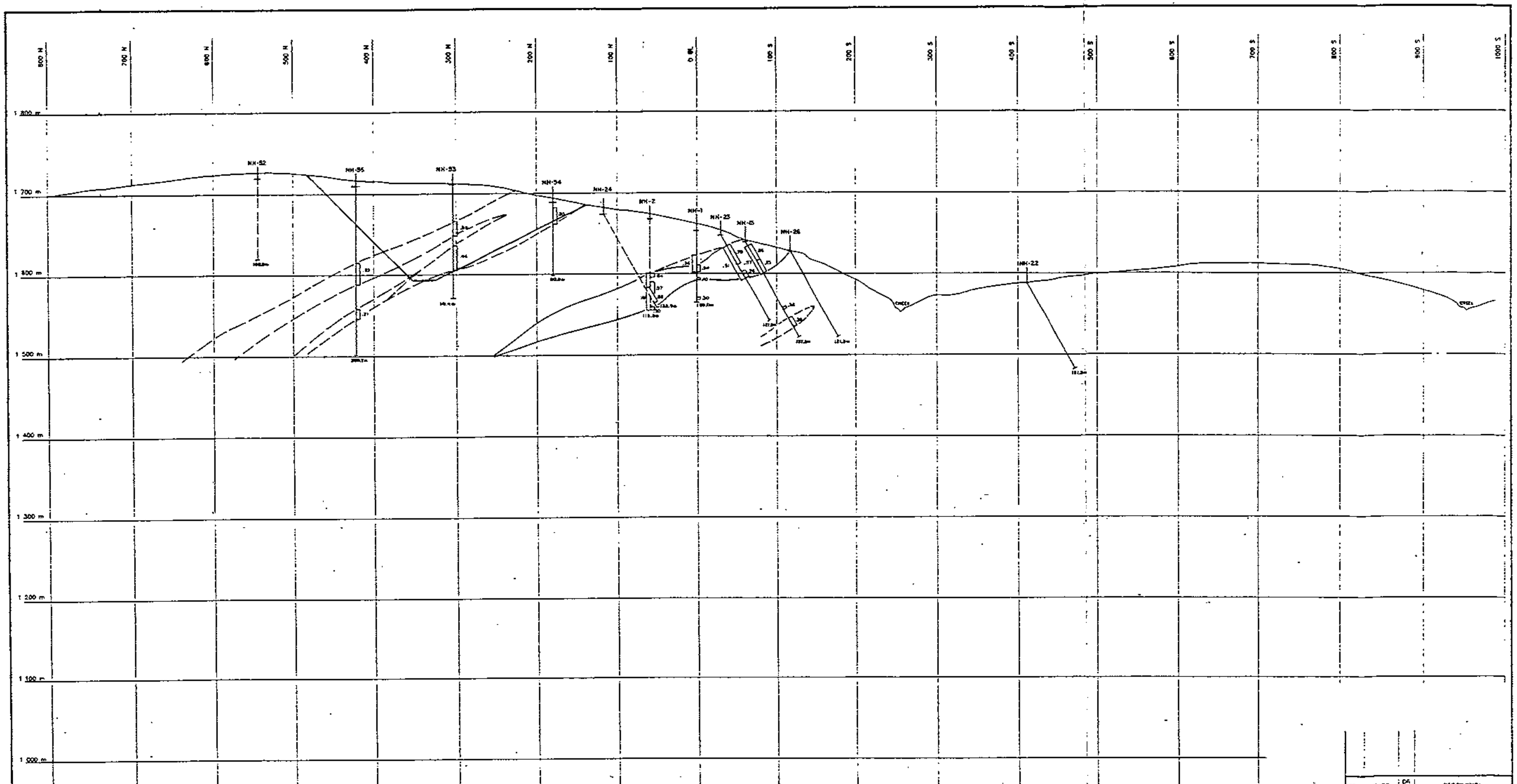
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1170W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV.
88005	HH043	

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

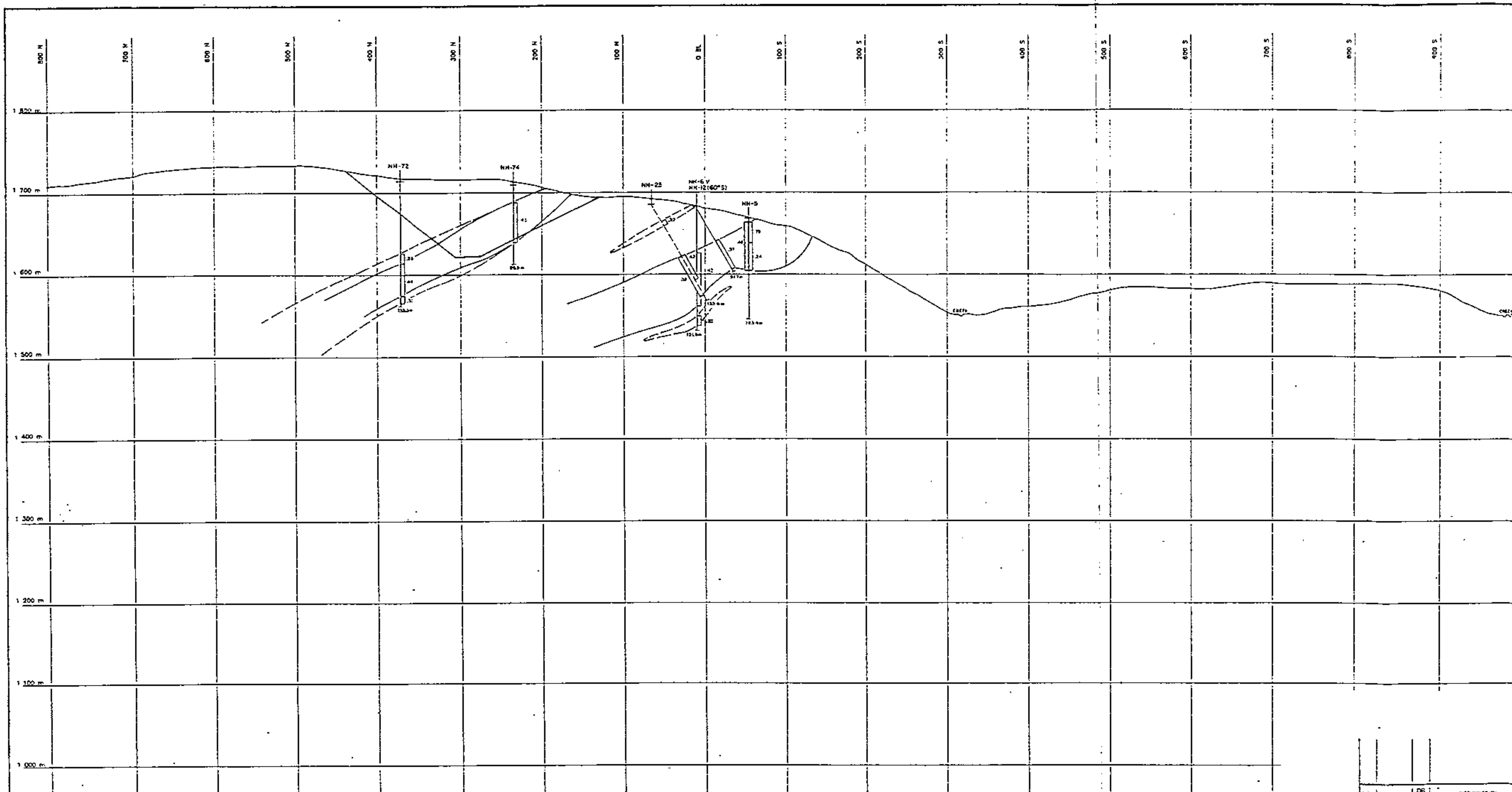
AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1260W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	
PROJECT No. 88005	DRAWING No. HH044	REV.

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV	DATE	DR / CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

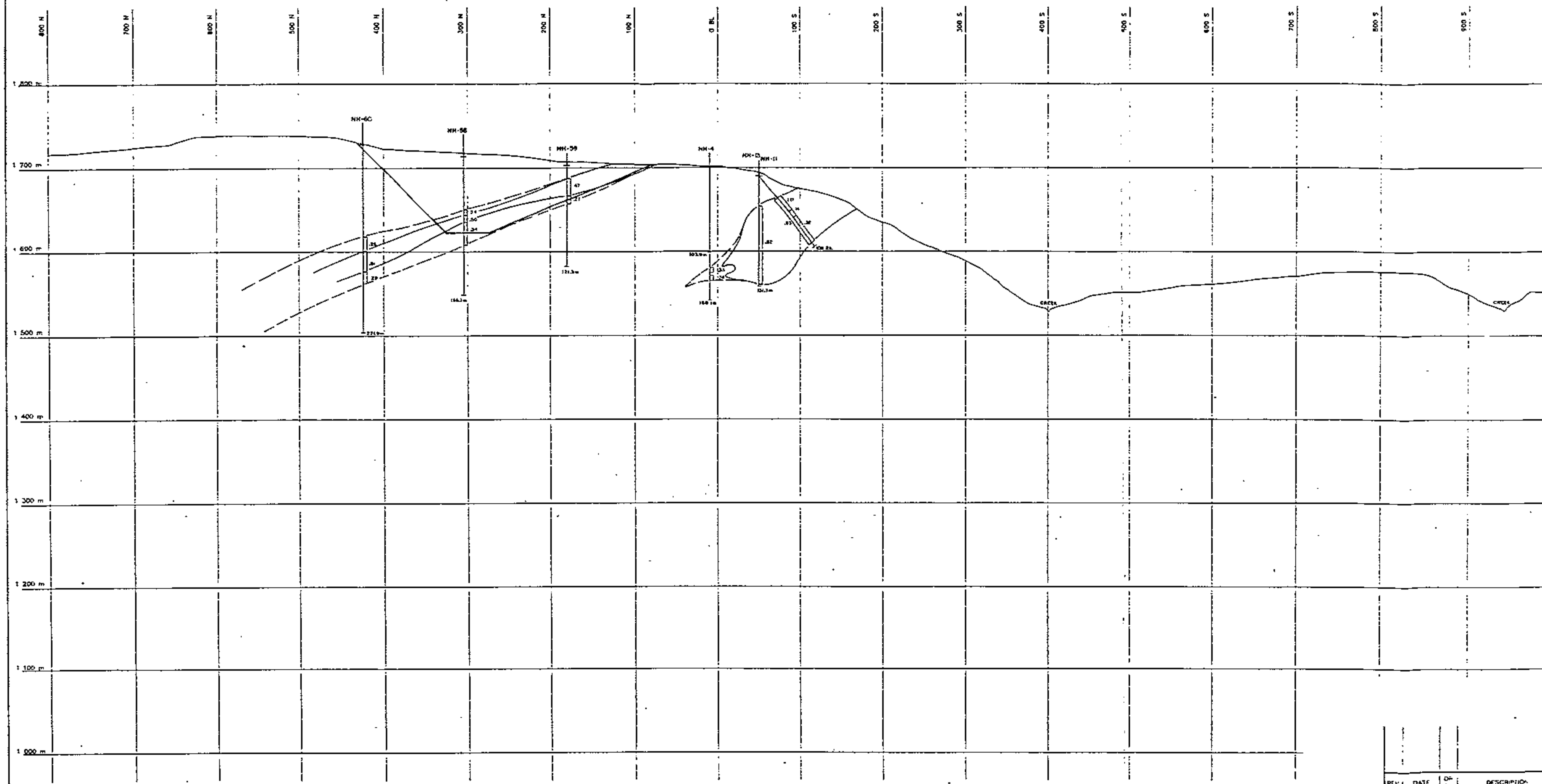
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1320W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1=2000	
DATE	88/03/30	

PROJECT No.	DRAWING No.	REV.
88005	HH045	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

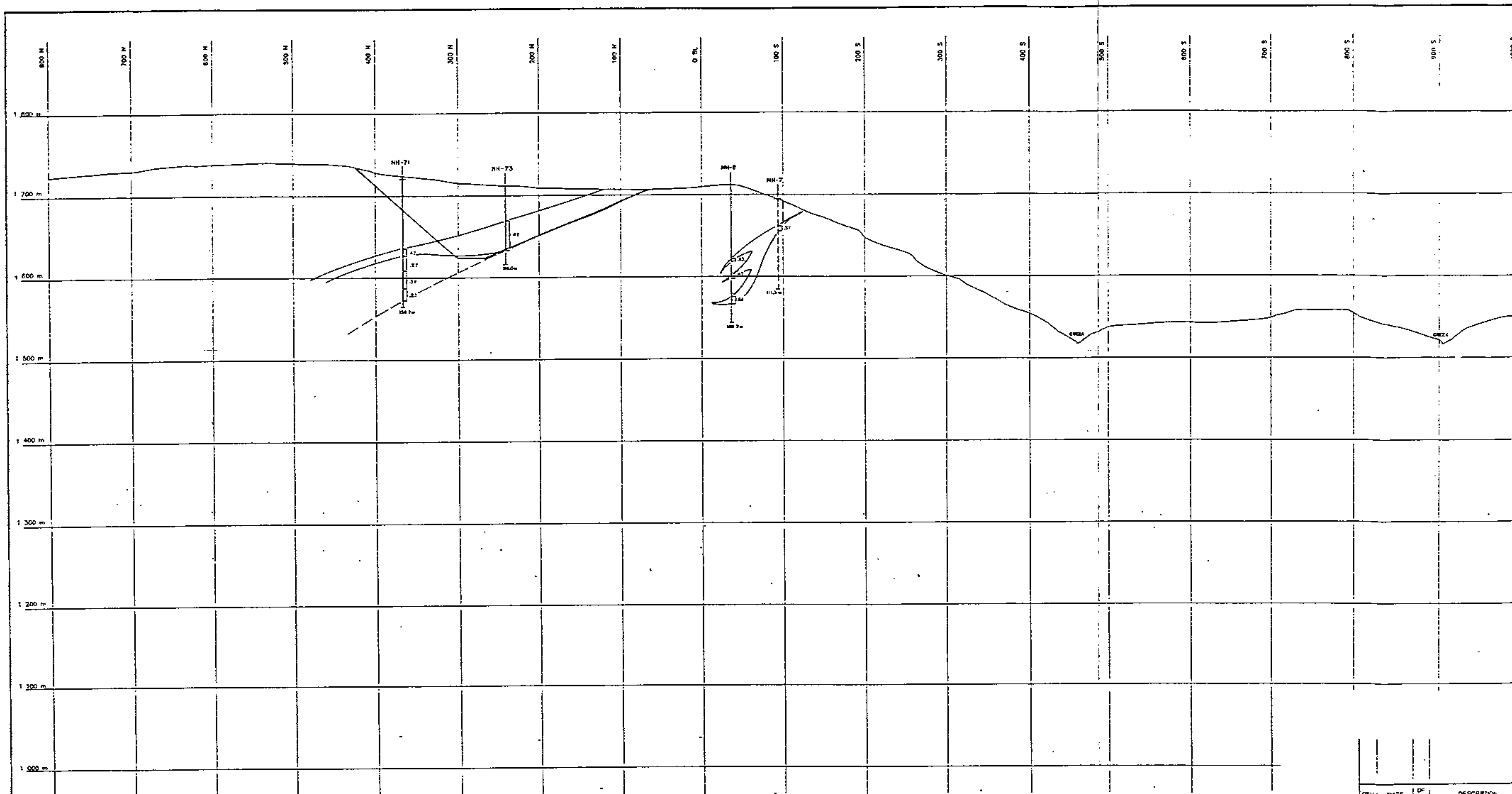
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1380W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	HH046	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DF / CH	DESCRIPTION

Phillips Barrett Kaiser
Engineering Ltd

AURUN MINES LTD.

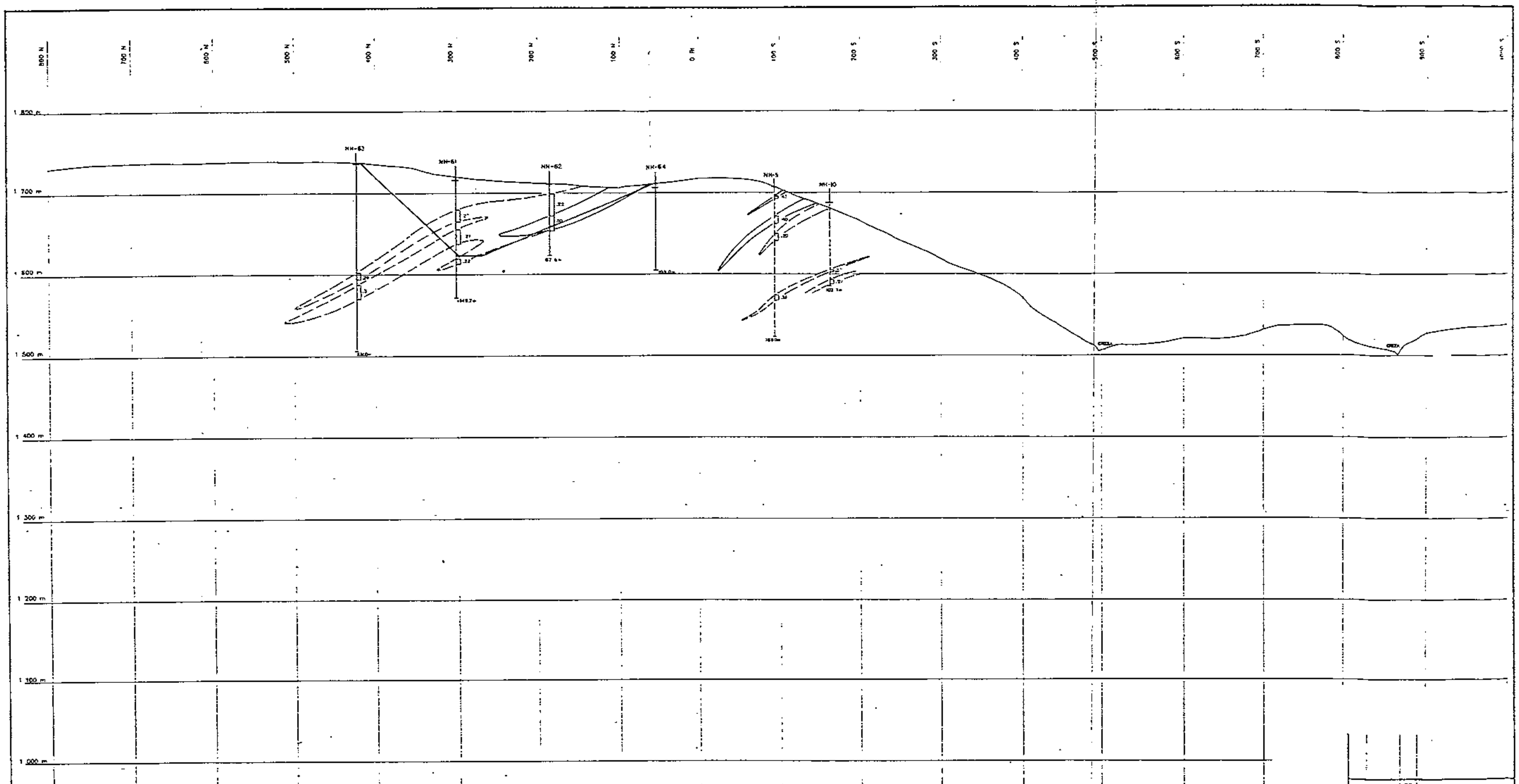
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1440W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT No	DRAWING No	REV
88005	HH047	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

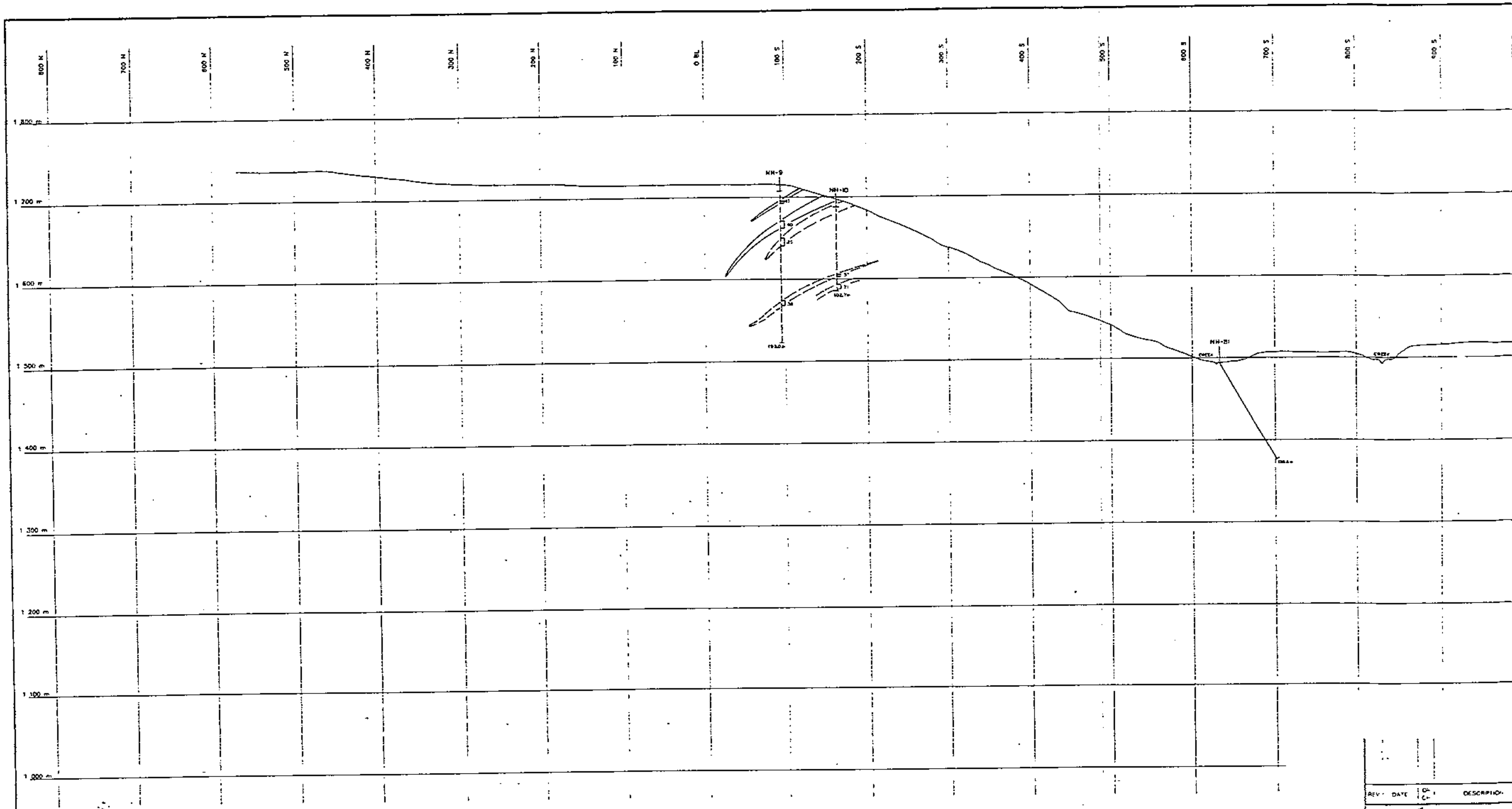
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1500W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT No.	DRAWING No.	REV.
88005	HH048	

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV	DATE		DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

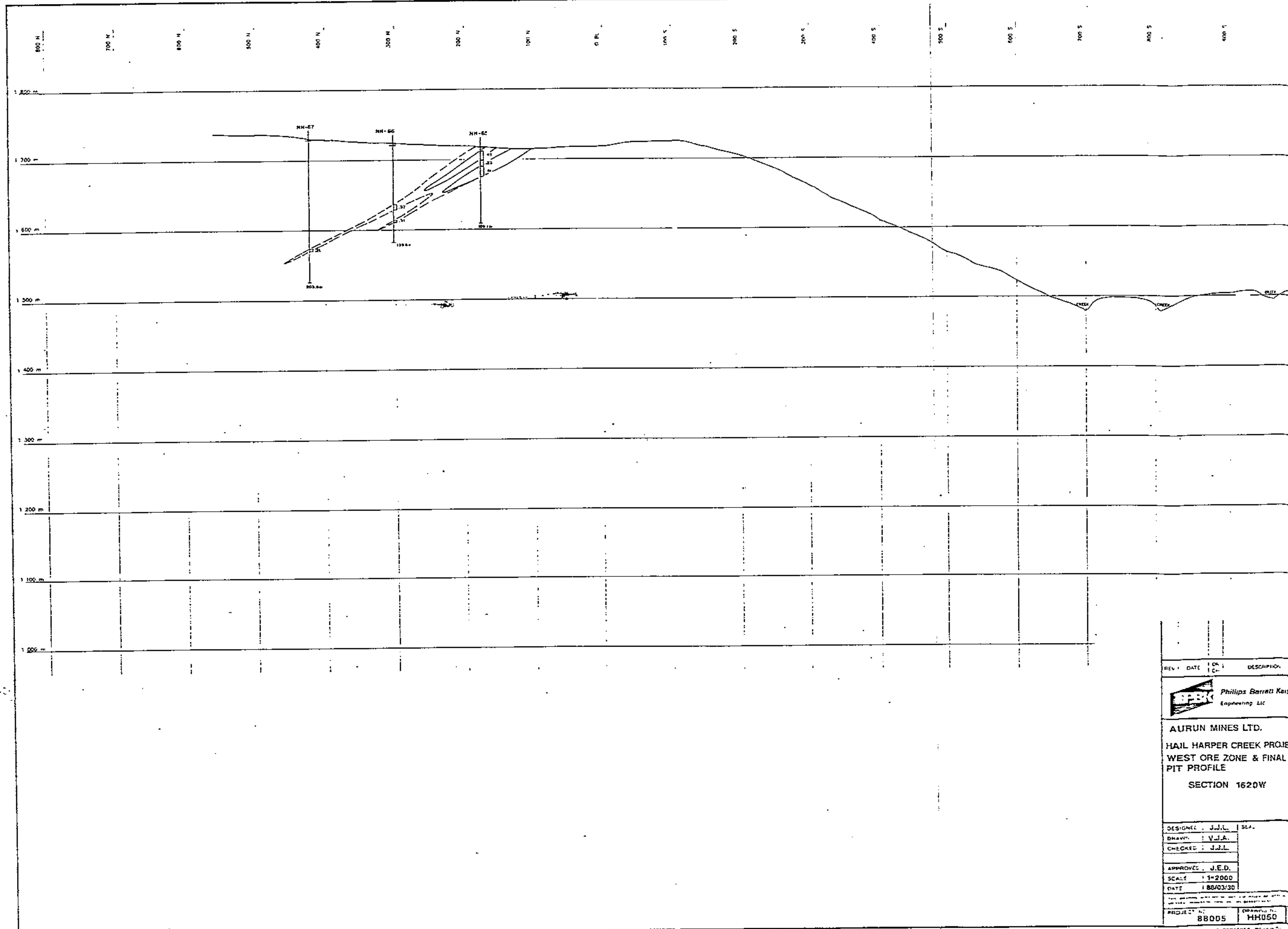
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL
PIT PROFILE

SECTION 1560W

DESIGNED	J.J.L.	SEAL
DRAWN	V.J.A.	
CHECKED	J.J.L.	
APPROVED	J.E.D.	
SCALE	1:2000	
DATE	88/03/30	

PROJECT No.	DRAWING No.	REV.
88005	HH049	

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV / DATE | CK / CH | DESCRIPTION

Phillips Barratt Kaiser
Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL
PIT PROFILE

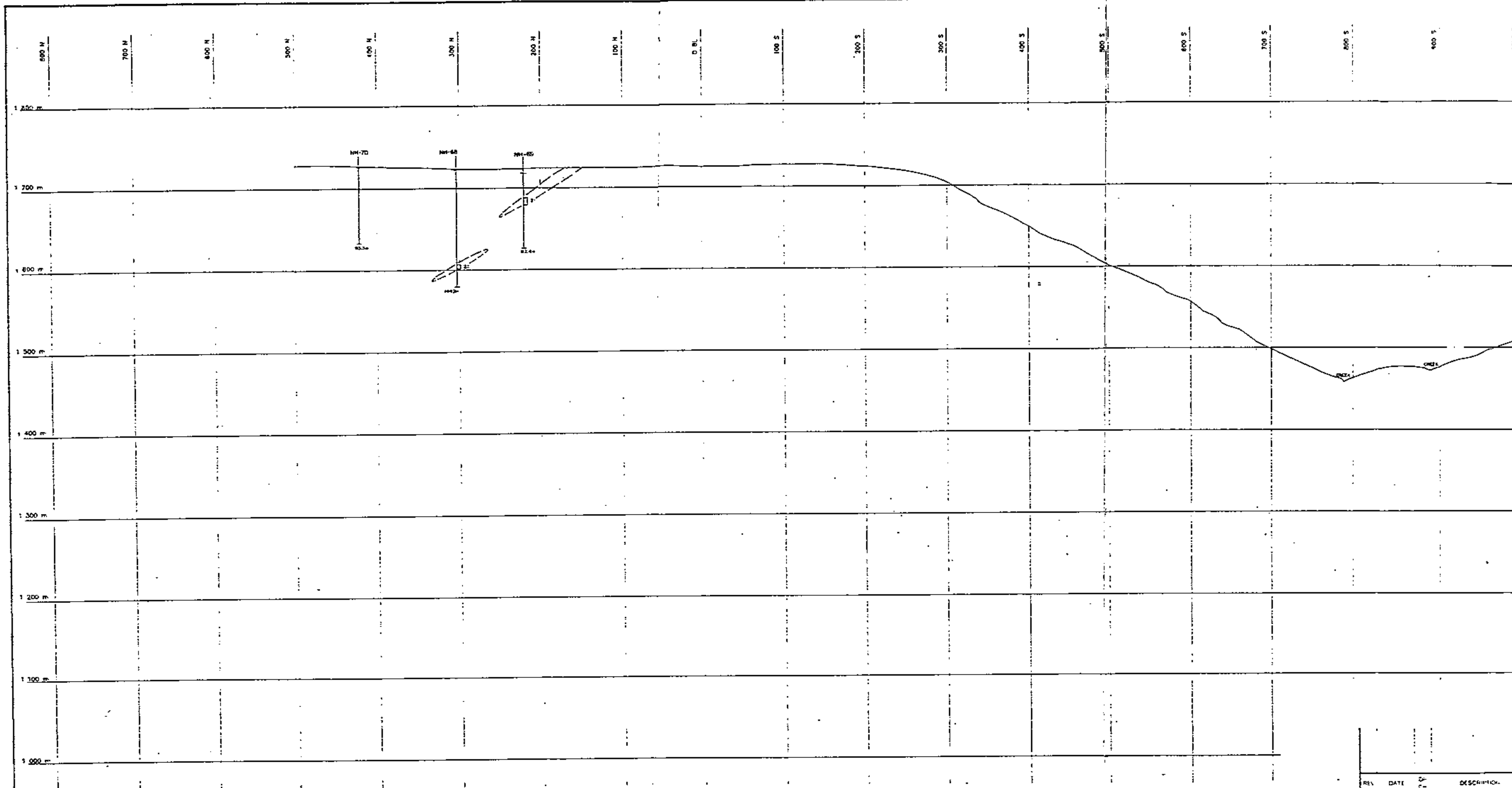
SECTION 1620W

DESIGNED	J.J.L.
DRAWN	V.J.A.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

SEAL

PROJECT No. 88005 | DRAWING No. HH050 | REV.

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



REV	DATE	CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.

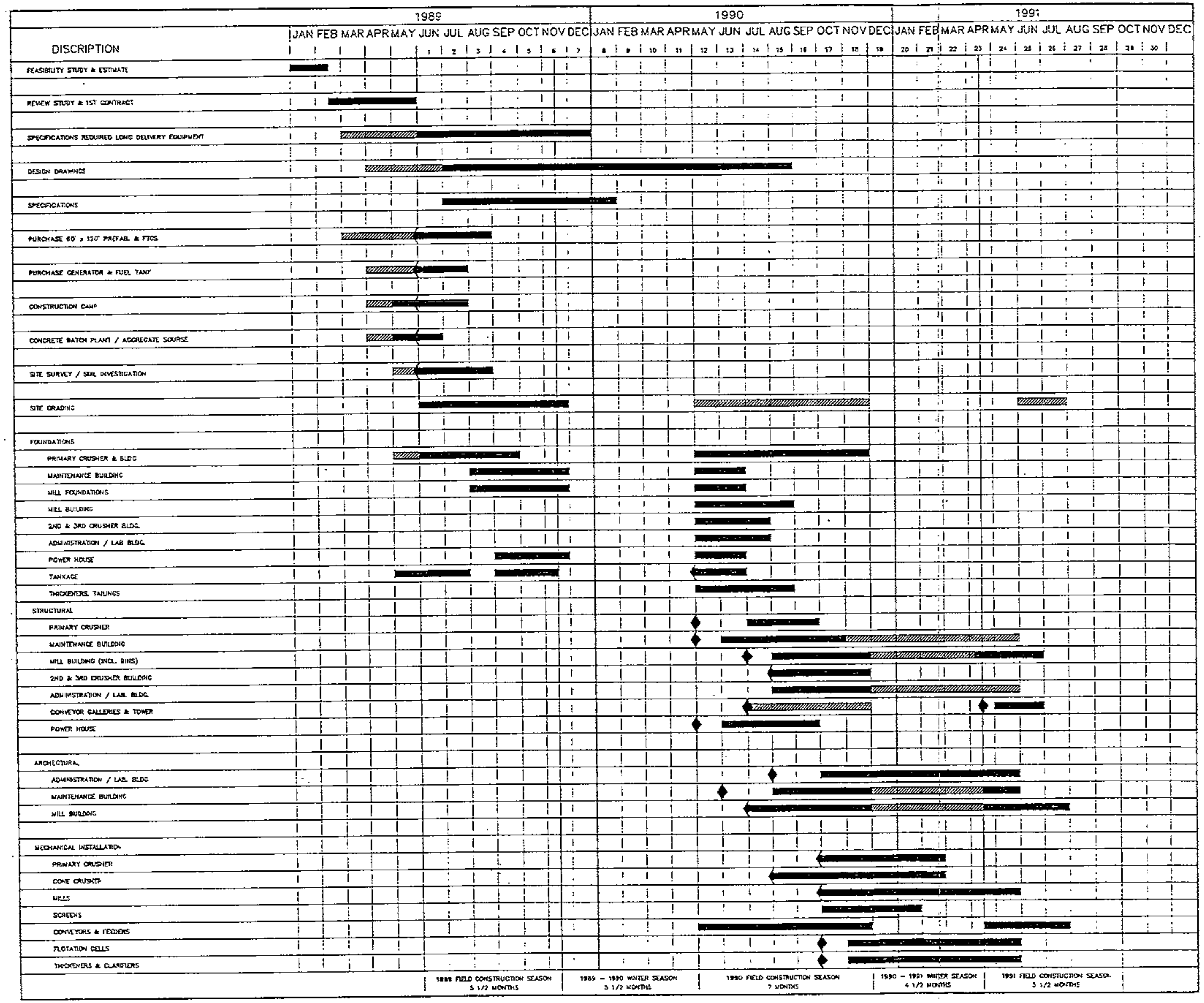
HAIL HARPER CREEK PROJECT
WEST ORE ZONE & FINAL PIT PROFILE

SECTION 1740W

DESIGNED	J.J.L.
DRAWN	V.J.A.
CHECKED	J.J.L.
APPROVED	J.E.D.
SCALE	1=2000
DATE	88/03/30

PROJECT No.	DRAWING No.
88005	HH051

THIS REVISION SUPERSEDES DRAWING BEARING PREVIOUS REVISION LETTER



NOTES

SCHEDULE IS BASED ON 28 MONTH PROJECT WITH A 10 MONTH WINTER SEASON AND 18 MONTH CONSTRUCTION PERIOD.

LEGEND

ENGINEERING / CONSTRUCTION

MOBILIZATION & INTERMITTENT CONSTRUCTION

DELIVERY OF EQUIPMENT TO SITE.

REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

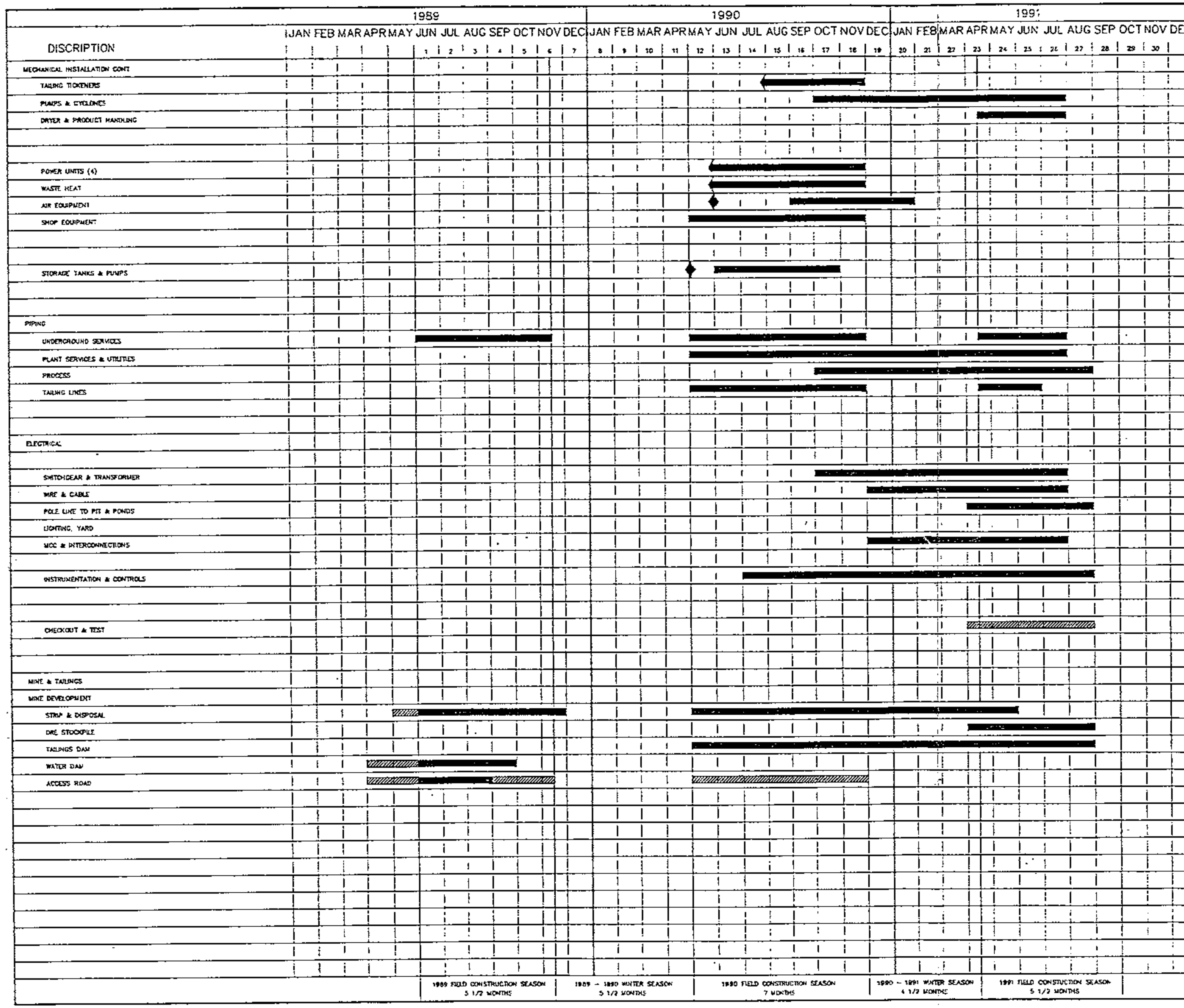
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

PROPOSED CONSTRUCTION SCHEDULE SHEET #1

DESIGNED		SEAL
DRAWN	J.R.	
CHECKED		
APPROVED		
SCALE	NONE	
DATE	30/03/88	

PROJECT NO.	DRAWING NO.	REV
88005	SH001	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



NOTES

SCHEDULE IS BASED ON 28 MONTH PROJECT WITH A 10 MONTH WINTER SEASON AND 18 MONTH CONSTRUCTION PERIOD

LEGEND

ENGINEERING / CONSTRUCTION

MOBILIZATION & INTERMITTENT CONSTRUCTION

DELIVERY OF EQUIPMENT TO SITE.

REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

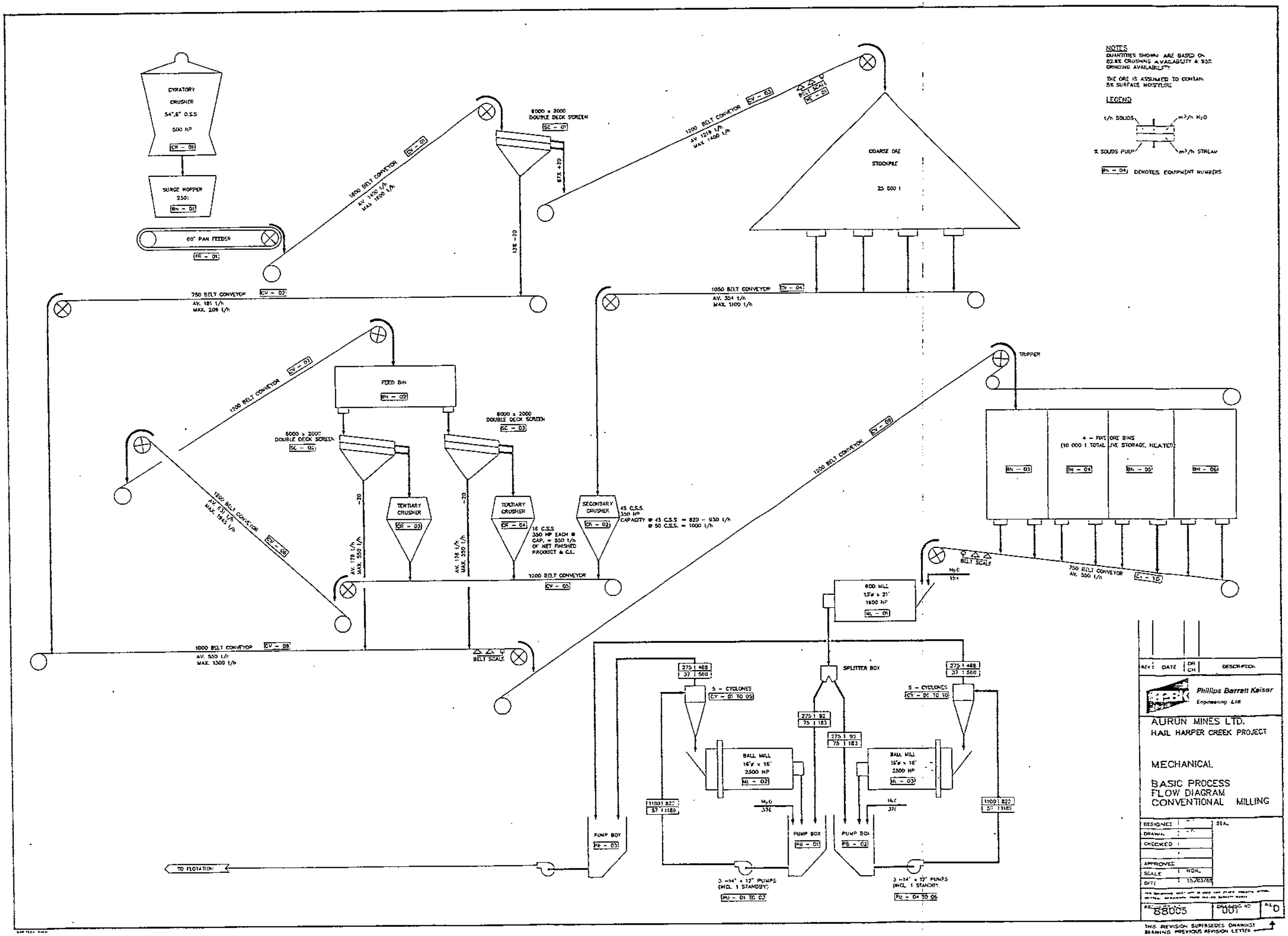
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

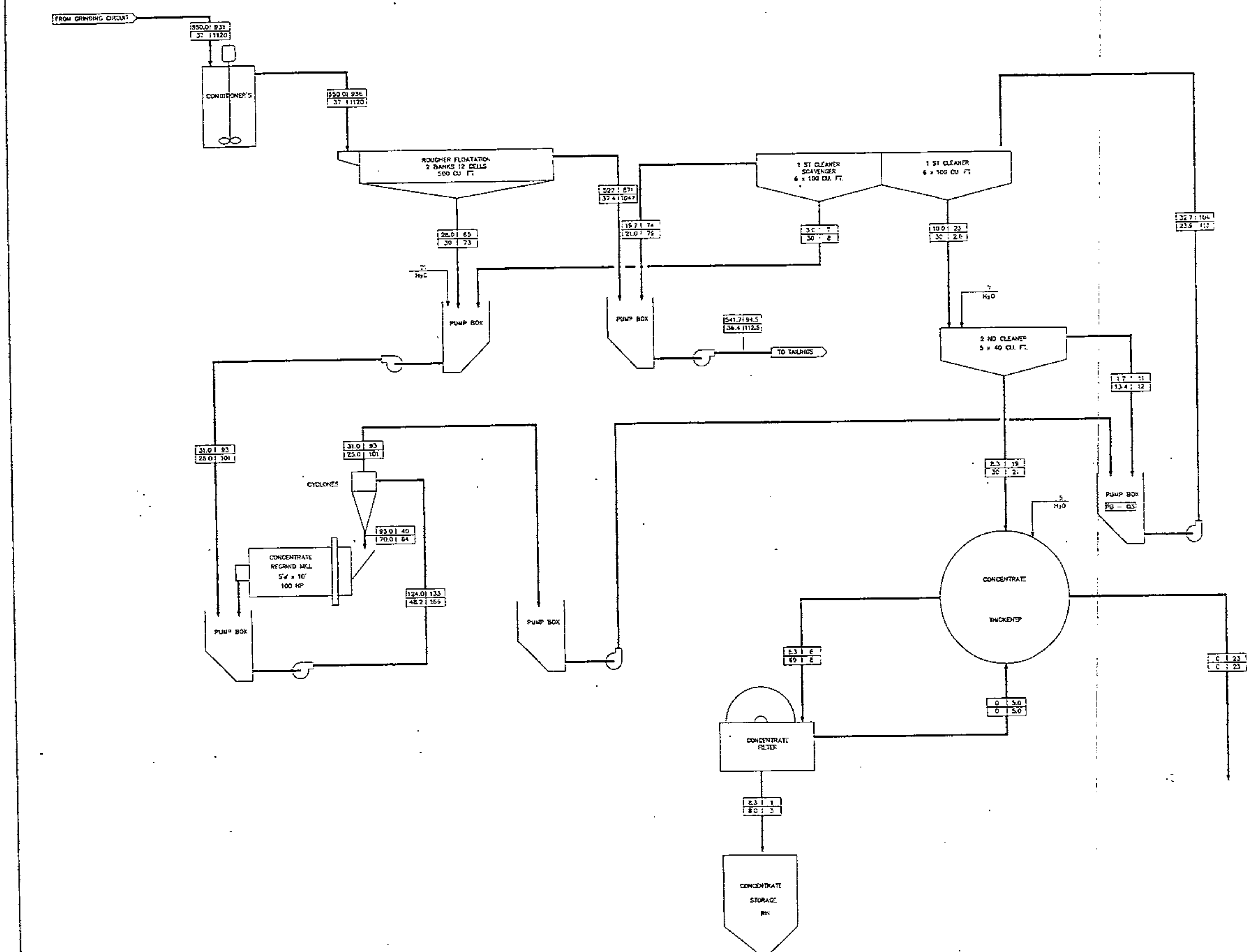
PROPOSED CONSTRUCTION SCHEDULE SHEET #2

DESIGNED	G.K	SEAL
DRAWN	J.F	
CHECKED		
APPROVED		
SCALE	NONE	
DATE	30/03/89	

PROJECT NO	DRAWING NO	REV
88005	SH002	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


NOTES
QUANTITIES SHOWN ARE BASED ON 82.8% CRUSHING AVAILABILITY & 95% GRINDING AVAILABILITY
THE ORE IS ASSUMED TO CONTAIN 5% SURFACE MOISTURE
LEGEND
t/h SOLIDS
m³/h H₂O
% SOLIDS PULP
m³/h STREAM
BN – 04 DENOTES EQUIPMENT NUMBERS
GYRATORY CRUSHER
54"x6" O.S.S
500 HP
CR – 01
SURGE HOPPER
250t
BN – 01
60" PAN FEEDER
FE – 01
1600 BELT CONVEYOR
CV – 01
AV. 1400 t/h
MAX. 1600 t/h
8000 x 2000 DOUBLE DECK SCREEN
SC – 01
87% +20
13% –20
1200 BELT CONVEYOR
CV – 03
AV. 1219 t/h
MAX. 1400 t/h
BELT SCALE
WE – 01
COARSE ORE STOCKPILE
25 000 t
750 BELT CONVEYOR
CV – 02
AV. 181 t/h
MAX. 209 t/h
1050 BELT CONVEYOR
CV – 04
AV. 354 t/h
MAX. 1100 t/h
1200 BELT CONVEYOR
CV – 07
FEED BIN
BN – 02
6000 x 2000 DOUBLE DECK SCREEN
SC – 02
8000 x 2000 DOUBLE DECK SCREEN
SC – 03
TERTIARY CRUSHER
CR – 03
TERTIARY CRUSHER
CR – 04
16 C.S.S
350 HP EACH @ CAP. = 550 t/h OF NET FINISHED PRODUCT & C.L.
SECONDARY CRUSHER
CR – 02
45 C.S.S.
350 HP
CAPACITY @ 45 C.S.S. = 820 – 930 t/h
@ 50 C.S.S. = 1000 t/h
1200 BELT CONVEYOR
CV – 06
AV. 631 t/h
MAX. 1845 t/h
AV. 178 t/h
MAX. 550 t/h
1200 BELT CONVEYOR
CV – 05
1000 BELT CONVEYOR
CV – 08
AV. 550 t/h
MAX. 1300 t/h
BELT SCALE
1200 BELT CONVEYOR
CV – 09
TRIPPER
4 – FINE ORE BINS
(10 000 t TOTAL LIVE STORAGE, HEATED)
BN – 03
BN – 04
BN – 05
BN – 06
BELT SCALE
750 BELT CONVEYOR
AV. 550 t/h
CV – 10
ROD MILL
13'ø x 21'
1600 HP
ML – 01
SPLITTER BOX
5 – CYCLONES
CY – 01 TO 05
5 – CYCLONES
CY – 06 TO 10
BALL MILL
16'ø x 16'
2500 HP
ML – 02
BALL MILL
16'ø x 16'
2500 HP
ML – 03
PUMP BOX
PB – 03
PUMP BOX
PB – 01
PUMP BOX
PB – 02
3 – 14" x 12" PUMPS
(INCL. 1 STANDBY)
PU – 01 TO 03
3 – 14" x 12" PUMPS
(INCL. 1 STANDBY)
PU – 04 TO 06
TO FLOTATION
REV DATE DR CH DESCRIPTION
Phillips Barrett Kaiser
Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
MECHANICAL
BASIC PROCESS
FLOW DIAGRAM
CONVENTIONAL MILLING
DESIGNED
DRAWN
CHECKED
APPROVED
SEAL
SCALE : NON.
DATE : 15/03/85
PROJECT NO 88005
DRAWING NO 001
REV D
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



NOTES
QUANTITIES SHOWN ARE BASED ON 82.5% CRUSHING AVAILABILITY & 95% GRINDING AVAILABILITY

THE ORE IS ASSUMED TO CONTAIN 5% SURFACE MOISTURE

LEGEND

t/h SOLIDS — m^3/h H_2O — % SOLIDS PULP — m^3/h STREAM

BN – 04 DENOTES EQUIPMENT NUMBERS

REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

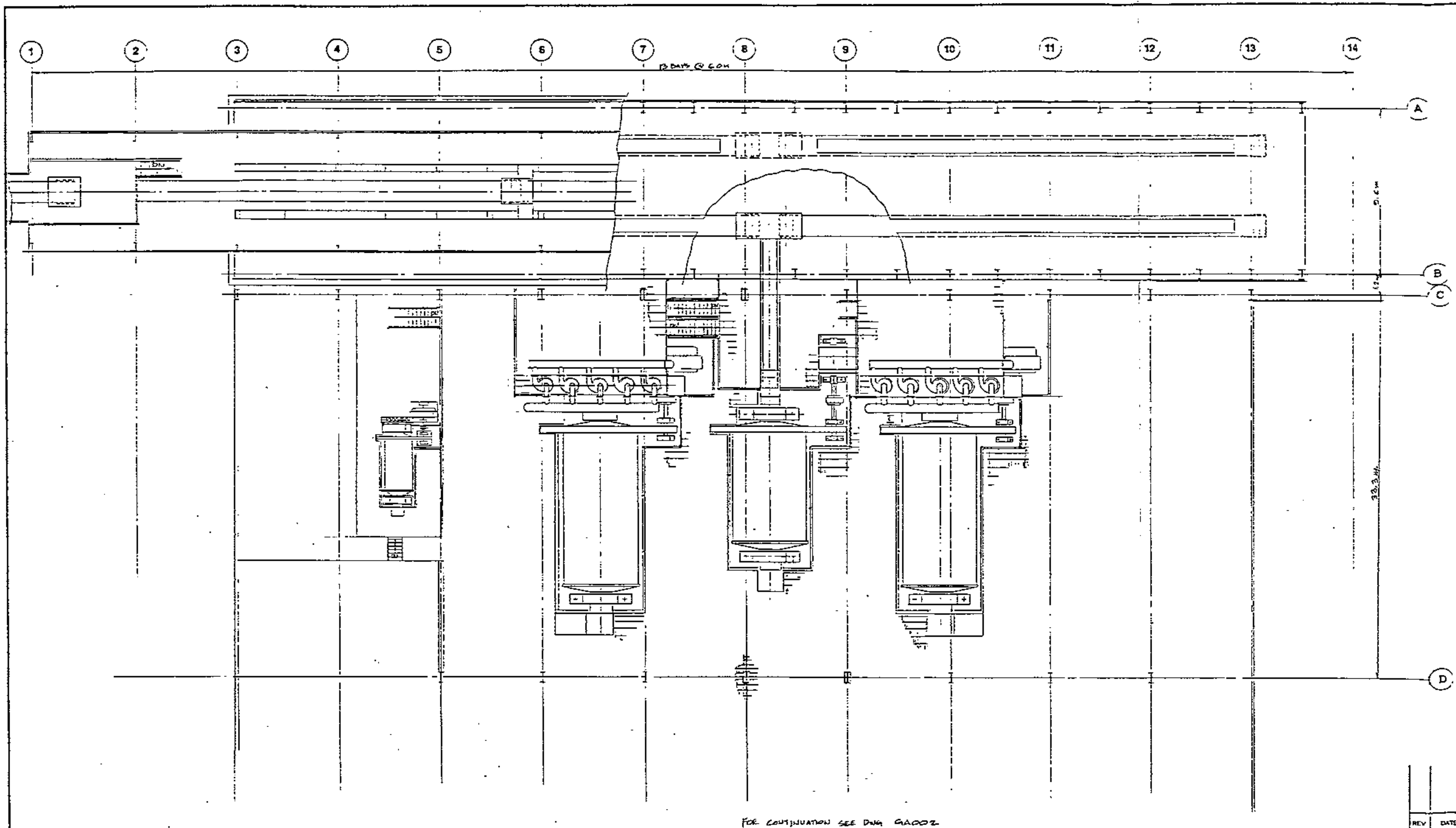
MECHANICAL

BASIC PROCESS
FLOW DIAGRAM
FLOTATION DEWATERING

DESIGNED		SEAL
DRAWN	J.R.	
CHECKED		
APPROVED		
SCALE	NONE	
DATE	30/03/85	

PROJECT NO.	DRAWING NO.	REV
88005	002	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barrett Kaiser Engineering Ltd

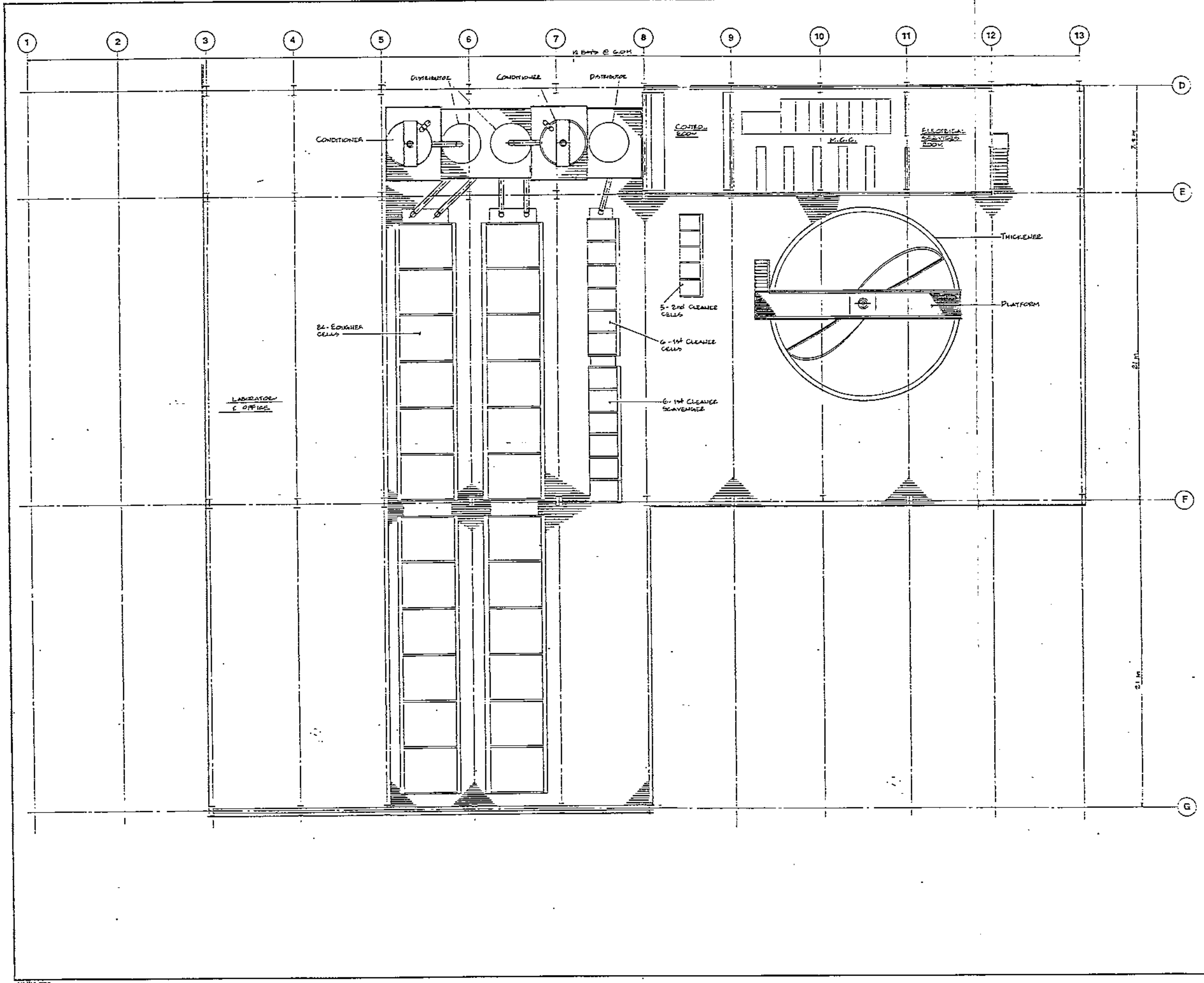
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

GRINDING PLANT LAYOUT

DESIGNED	J.R.	SEAL
DRAWN	J.R.	
CHECKED	G.W.	
APPROVED	J.E.D.	
SCALE	1=100	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	GA001	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barrett Kaiser Engineering Ltd

AURUN MINES LTD.

HAIL HARPER CREEK PROJECT

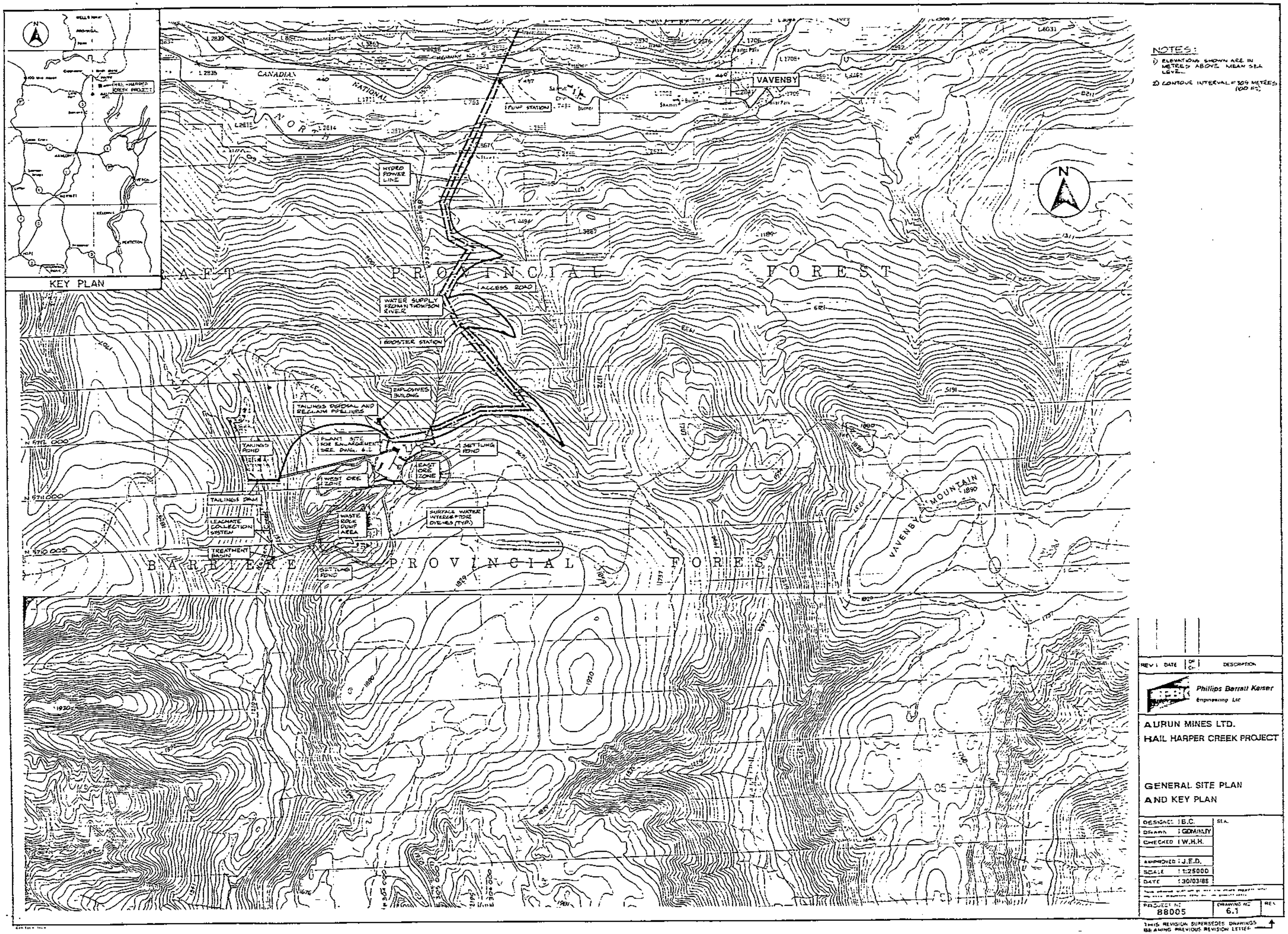
FLOTATION PLANT

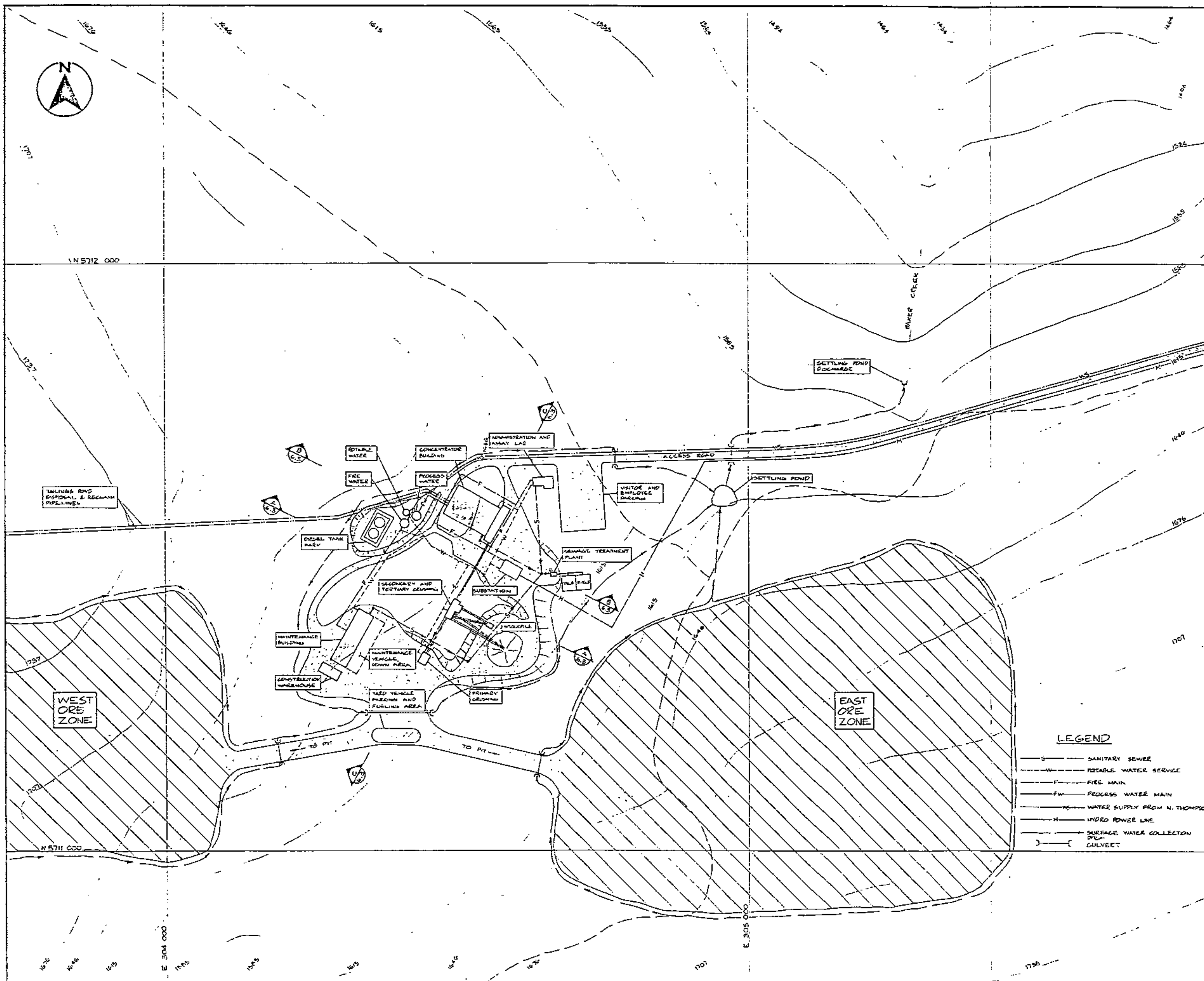
LAYOUT

DESIGNED	J.R.	SEAL
DRAWN	V.J.A.	
CHECKED	G.W.	
	–	
APPROVED	J.E.D.	
SCALE	1=100	
DATE	88/03/30	

PROJECT NO	DRAWING NO	REV
88005	GA002	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER


KEY PLAN
HAIL-HARPER CREEK PROJECT
VAVENBY
CANADIAN NATIONAL
FUMP STATION
HYDRO POWER LINE
ACCESS 2040
WATER SUPPLY FROM N. THOMPSON RIVER
BOOSTER STATION
EXPLOSIVES BUILDING
TAILINGS DISPOSAL AND RECLAIM PIPELINES
PLANT SITE FOR ENLARGEMENT SEE DWG. 6.2
TAILINGS POND
SETTLING POND
EAST ORE ZONE
WEST ORE ZONE
TAILINGS DAM
LEACHATE COLLECTION SYSTEM
WASTE ROCK DUMP AREA
SURFACE WATER INTERCEPTOR DITCHES (TYP.)
TREATMENT BASIN
SETTLING POND
PROVINCIAL FOREST
BARRIERE PROVINCIAL FOREST
VAVENBY MOUNTAIN
N
NOTES:
1) ELEVATIONS SHOWN ARE IN METRES ABOVE MEAN SEA LEVEL.
2) CONTOUR INTERVAL = 30.9 METRES (100 FT)
REV | DATE | DF CH | DESCRIPTION
Phillips Barratt Kaiser Engineering Ltd
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT
GENERAL SITE PLAN
AND KEY PLAN
DESIGNED : B.C.
DRAWN : GDM/MJY
CHECKED : W.H.H.
APPROVED : J.E.D.
SCALE : 1:25000
DATE : 30/03/85
SEAL
PROJECT NO 88005
DRAWING NO 6.1
REV
THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

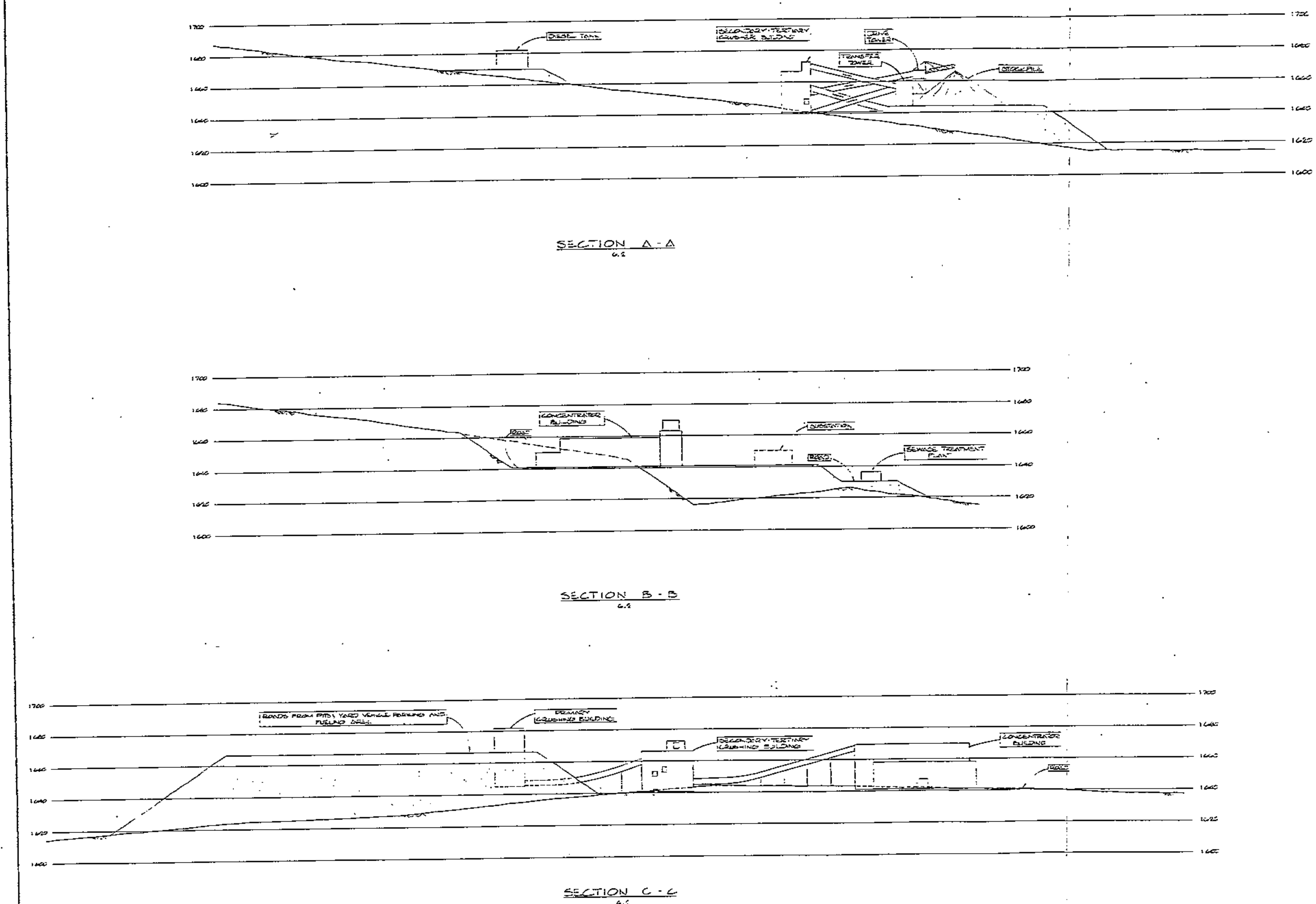
AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

SITE FACILITIES AND YARD UTILITIES

DESIGNED	B.C.	SEAL
DRAWN	GDM/MJY	
CHECKED	W.H.H.	
APPROVED	J.E.D.	
SCALE	1:2500	
DATE	30/03/88	

PROJECT NO	DRAWING NO	REV
88005	6.2	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV.	DATE	DR CH	DESCRIPTION

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

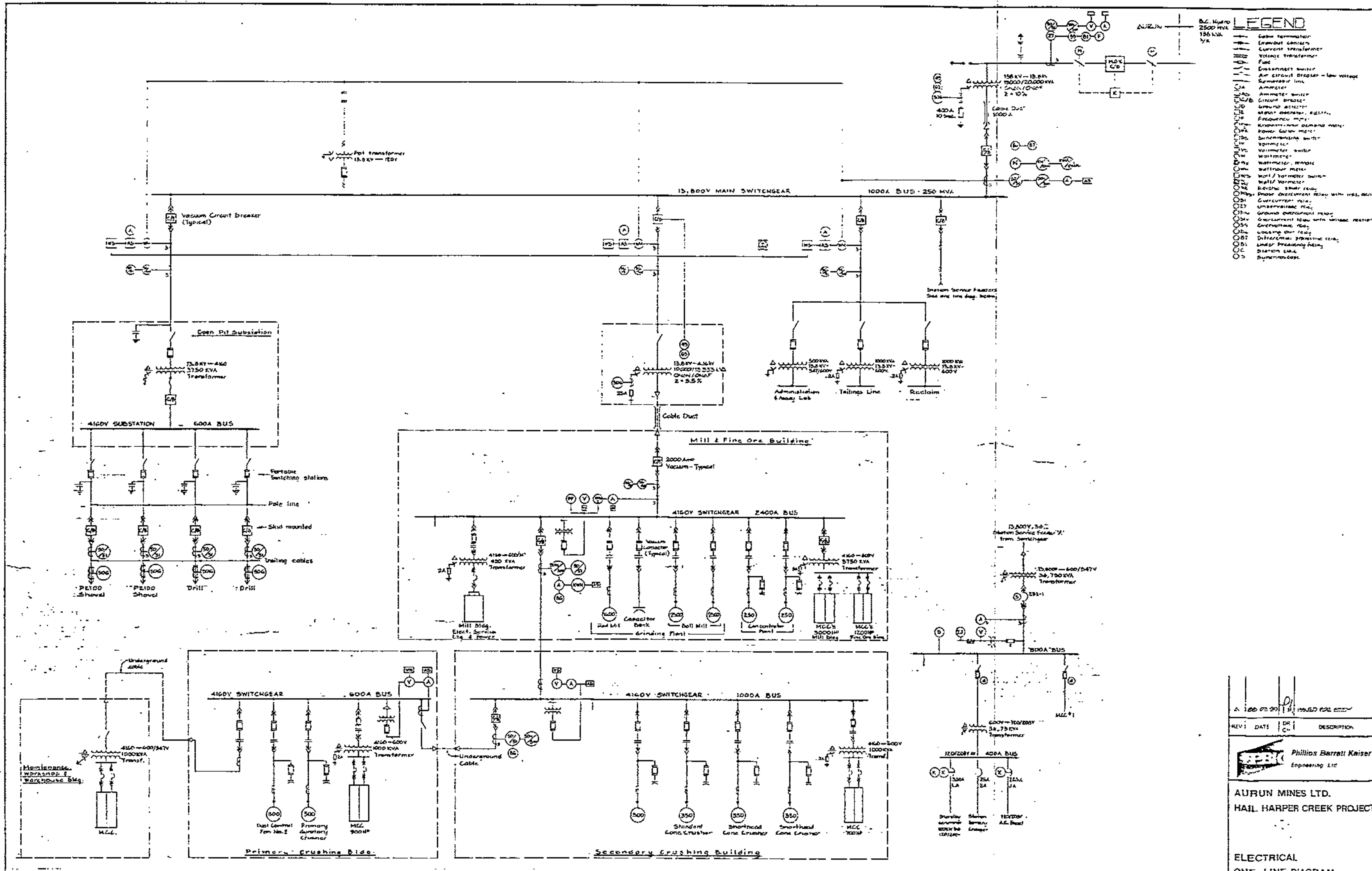
SITE FACILITIES
SECTIONS

DESIGNED	B.C.
DRAWN	D.M.B.
CHECKED	W.H.H.
APPROVED	J.E.D.
SCALE	1:1000
DATE	30/03/88

SEAL

PROJECT No.	DRAWING No.	REV
88005	6.3	

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER



REV	DATE	DR CH	DESCRIPTION
A	88 03 29		ISSUED FOR STUDY

Phillips Barratt Kaiser Engineering Ltd

AURUN MINES LTD.
HAIL HARPER CREEK PROJECT

ELECTRICAL
ONE-LINE DIAGRAM

DESIGNED	I.M.S.
DRAWN	G.P.
CHECKED	I.M.S.
APPROVED	J.E.D.
SCALE	
DATE	30/03/85

SEAL

PROJECT No. 88005 | DRAWING No. 6.4 | REV

THIS REVISION SUPERSEDES DRAWINGS BEARING PREVIOUS REVISION LETTER