SFK - 378 - 5513

812650

Bralorne Pioneer Mines, Ltd. ← File

Bralorne, B.C.

SPHALER Ck
KENNCO - 80% -
① 1962-63 staked
② Zone - 3 miles N-S
③ Explor. Targets are FRACTURED qtz-mon Company caps

LORNOX
.04 Mo
.4.5 Cu
~~Mostly~~ Bornite center
~~Cu~~ py N & S
No py

VAN. OFF -

Y-1 -

RECENT DEVELOPMENTS AT THE BRALORNE MINE

J.J. Thompson,
Chief Engineer,
Bralorne Pioneer Mines Ltd.,
Bralorne, B. C.

Presented at:
CIMM B.C. Section & Victoria
Branch Convention, Victoria, B.C.
October 29th, 1966

Introduction:

The purpose of this paper is to summarize the following recent developments at the Bralorne mine.

(1) Insulation as applied to mining in rock of high temperatures.

(2) Raising by rotary boring methods.

(3) Auto reagent feeding as applied to the mill cyanidation circuit.

General:

Many of you present will be familiar with the Bralorne gold mining operation. The mine is located at an elevation of approximately 3,500' in the Coast Range mountains and 110 miles north of Vancouver. Mining is from a depth of 5,800 ft. through three shafts and lateral development totals over 55 miles in drifts and crosscuts. The gold bearing quartz ore is extracted by cut and fill stoping over an average width of 8' and dip of 80°. Recovery is by amalgamation and cyanidation, the mill tailings being classified and returned underground as hydraulic backfill.

Heat Insulation:

One of the major problems associated with mining at Bralorne is the maintenance of satisfactory environmental conditions in rock of high ambient temperature. Fig. (1) shows the geothermal gradients of certain mining localities and indicates the root of our problem. The geothermal gradient for Bralorne exhibits a 1°F rise in temperature for every 67.5 ft. in depth, three times greater than the gradient for, example, Timmins, Ontario. As a result, at the lowest level of the mine, the 43rd, 5,800 ft. below surface, the rock temperature is 128°F.

It is relevant to note that at the moment 100% of the mine production is extracted from rock of a temperature greater than 100°F; 95% is extracted from rock of a temperature greater than 110°F, and 25% is extracted from rock of a temperature greater than 120°F.

For all working places it is desirable that we maintain the wet bulb temperature no higher than $83^{\circ}F$. An addition to the problem is combatting high relative humidities which are caused, in large part, by the use of hydraulic mill tailings as backfill in the stopes.

The mine ventilation is essentially a forcing system based on two 48" diam. Jeffrey aerodyne fans, located on surface, and blowing 100,000 cfm at $7\frac{1}{2}$" w.g. through vent raises to the collar of the Queen shaft. Booster fans located on the lower levels draw fresh air from the Queen shaft for distribution to the working areas. The air then exhausts through the stopes and to surface via the Crown and Empire shafts. Any heat transferred to the air on the lowest levels thus has an adverse effect on the working areas above.

In attempts to maintain satisfactory working conditions consideration has been given to several schemes. Firstly, and foremost, a revamping of the entire vent circuit, including an extension to the vent raise, is indicated in order to provide greater volumes of cool air. Although certain improvements have been carried out to the exhaust circuit an overall program would involve a large expenditure in manpower, time, and money.

On the lower levels, therefore, with a limited supply of cool fresh air available from the Queen shaft the problem has become one of directing this air to the working places without an undue rise in temperature. Consideration has been given to the use of refrigeration units and also cooling towers or spray chambers. These methods have proved effective in other hot mines, but in addition to high capital cost also require a heat transfer medium, usually cooling water, and power supply, which in our case are not immediately available.

In 1963 the problem was approached from a different angle: that of preventing the heat transfer from the rock to the air rather than trying to cool the air after this heat transfer had taken place. Consideration was therefore given to the insulation of the walls and backs of drifts and cross-cuts with sprayed Urethane foam; or ventilating through insulated fan ducting. At that time the equipment for spraying rigid Urethane foam did not seem to be readily available. However the idea of insulated vent ducting matured and has been a complete success. It is no exaggeration to state that without it development of the 39, 41, 42 and 43 levels would have been almost impossible. The duct as designed by Mr. W. E. Field, now Resident Manager, consists of an inner lining of 15 mil poly-tubing 22" ID, a middle layer of 1" fibreglass, and an outer cover of waterproof mildew resistant canvas. The heat transfer coefficient K is 0.21 BTUs/sq. ft/hr/^{O}F. In theory 4,000 cfm of air at a temperature of 77^{O} d.b. could be blown through 2,500' of this duct situated in an atmosphere of 95^{O}F (Sat.) with an increase in temperature of only 3^{O}F d.b. In practice the temperature increases have been up to double the theoretical due mainly to imperfections in the manufacture and installation of the tubing.

In developing the 43rd level cross-cut in an ambient temperature of 128^{O}F it was found impossible to maintain reasonable working conditions even with insulated vent duct. In mid 1965 a portable urethane foam spraying system was leased and considerable experimentation carried out into the insulation, sealing, and adhesion properties of rigid urethane foam. These tests proved to be very promising and early in 1966 it was decided to insulate the walls and back of the 43 level cross-cut, then advancing to an ultimate length of 900'. Ideally the insulation should have been maintained within 30' of the face but due to delays caused by equipment and supplies the foaming often lagged by 100 ft. to 300 ft. Nevertheless with the heading up to 60% insulated the heat transfer to the return air was significantly reduced.

In Fig. 2 the temperatures of the exhaust air from two different development headings are illustrated graphically. The solid line represents the 38 level Pioneer cross-cut, driven in 1964 in 116^{O}F rock, using insulated duct but no insulation on the walls. The dotted line represents the 43rd level being driven in rock at 128^{O}F using insulated duct and urethane foam sprayed to within 300' of the face. The return air from the Pioneer crosscut shows a temperature gradient which levels off at a point approximately 900' back from the face at a temperature of 97^{O} d.b. In the 43rd level cross-cut the temperature rise is much steeper due to the higher rock temperature and reaches 96^{O}F in only 300' from the face where it enters the section of crosscut which has been insulated. A drop of 1^{O}F is shown due to cool air leaking from a joint in the vent duct, and this temperature is maintained for the remaining 400 feet to the shaft station.

The foam spraying system, designated Urefroth 202 is supplied by Polymir Industries of Richmond, California. The equipment is self contained and consists of a spray gun and one tank each of pre-foamed resin, pre-foamed catalyst flushing solvent and nitrogen. The nitrogen pressurizes the other three tanks which are connected to the gun by hoses. The resin and catalyst are proportioned in a 1:1 weight ratio by factory pre-set metering controllers and pass through a pre-expansion and mixing chamber before leaving the gun. Upon contact with the sprayed surface a further expansion of about 100% occurs.

The sprayed material becomes tack free within a minute or so forming an 80% closed cell structure. It has a core density of 1.8 lbs/ft 3 and a 1 inch thickness has a heat transfer coefficient (K) of 0.15 BTU/sq. ft/hr/^{O}F mean temperature difference. Adhesion is generally excellent providing that the surface is fairly dry and dust free.

To date over 30,000 board feet of foam have been applied underground for insulation purposes. A small amount has also been used for vent bulkheads and stoppings. The average cost per board foot has been approximately 35¢, although as operator experience increased this figure improved.

Full assessment of this program will not be possible until through ventilation is established on the 43rd level. However the indications are that the experiment will be successful and may provide valuable information in the event of mining at greater depths.

Raise Boring:

The driving of raises at Bralorne is subject to most of the difficulties experienced in similar mines. Of late, however, the problem of high rock temperatures and lagging development have become a prime concern.

In order to establish an effective ventilation and ore transfer system, a total of over 1800 feet of raising is required within a six month period. By conventional hand raising or raise climber techniques, this is beyond the capabilities of our operation.

In late 1965, our staff decided to explore the possibility of raise boring as a possible solution. Following examinations of the different raise borers at other underground operations, the decision was made to use the raise borer designed by J.S. Robbins & Associates of Seattle.

It was noted that all operators, who were using, or had had experience with the raise borer, were most enthusiastic as to its possibilities.

The application of rotary boring techniques in the mining industry is not new, and considering the publicity of late most of you will be familiar with the principles involved in raise boring. The method advanced by Robbins and Associates is that of drilling a pilot hole, (usually 9-7/8" diam.), down from the upper location to the target below where the pilot bit is detached and replaced by a reamer head of requisite diameter designed for up-drilling. The thrust applied on the down hole is then reversed and the hole reamed up to the collar. There are several variations on the principle of pilot hole and reaming but the above method appears to have been the most satisfactory to date.

It was felt that the Robbins 41 R Raise Borer had additional advantages over other designs. In addition to its compact dimensions the machine could be broken down easily for transportation in 4' x 5' shaft compartments, and the heaviest component weighs only two tons. The maximum power requirements are in the region of 100 H.P. when reaming, which is just within the spare H.P. limits available on the lower levels underground.

The machine, components, part of the drill stem and the Robbins demonstration crew arrived on the property on Sept. 13. By Sept. 17th the machine was reassembled on the 38 level and anchored to the collar of the first hole, a length of 130' at 82° dip. The sequence of events was then as follows:

Sept.	17	8:45 a.m.	9-7/8" pilot hole collared.
"	17	11:30 p.m.	136' drilled. Bit located a few inches in the hangingwall 39 level, approximately 4' off target (layout error).
"	18		Hangingwall slashed to accommodate 4 foot diameter reamer.
"	19	6:35 p.m.	Reaming commenced.
"	21	2:40 a.m.	Reaming completed.

The penetration rates particularly for the pilot hole are considered to be fairly high. The rock type is a variable diorite and quartz with an ultimate compressive strength in the region of 20,000 p.s.i.

Two more raises have been drilled to date, one for a length of 240' and one for 344'. Relevant data for all three 48" raises is tabulated below:

Raise No.	Length	Dip	Rock - Type	Penetration Rate Pilot	Reamer	Elapsed Time Collar-Collar	Accuracy
1	130'	82°	Qtz-Diorite	14.5ft/hr	5.3ft/hr	90 hrs.	4' off target
2	240'	82½°	" "	10.2 "	3.9 "	132 "	On target
3	344'	83°	" "	12.9 "			

The only major problem encountered was excessive cutter wear during the reaming cycle of the 240' hole. The net result was that the reamer could only be pulled to within 15' of the collar thus making it necessary to blast the remaining footage. The cause of such unusually high cutter wear was probably due to a combination of several factors which will not be discussed at this stage.

It is probably true to state that this raise boring demonstration at Bralorne has been very successful and is clearly illustrating the tremendous advantages of the method.

Automatic Reagent Feeding

In the metallurgical field instrumentation is being developed for an increasing variety of process controls. Because of the static price of gold it is increasingly necessary to cut costs and increase efficiencies. The question that may be asked is, "Can process control by instrumentation do either of these."

The Department of Energy, Mines and Resources has recently developed two controls for use in the cyanide circuits of Canadian gold mills. These are the

Alkalinity Probe and the Continuous Cyanide Titrator both of which are discussed in the paper 'Instrumentation in the Cyanidation Process'.[I]

Upon request these process controls and automatic recording equipment were installed in the Bralorne mill by two scientific officers from the Department of Energy, Mines and Resources. The object was to determine if the equipment was amenable and applicable to this particular circuit.

The alkalinity probe consists of a primary coil with 12 turns of wire and a secondary coil of 5,000 turns. These are mounted in a stainless steel container and insulated from each other with silicone rubber. The wall of the 1/2" bore through the container is made of an insulating material such as Teflon. The probe is immersed in the pulp and a sinusoidal voltage of 1000 c/s applied to the primary coil. The secondary coil thus experiences an induced sinusoidal voltage proportional to the conductivity of the pulp. This is recorded on a chart calibrated to give a direct reading of alkalinity. For automatic control of lime addition a set point is established on the chart and any variation from this point is sensed by a variable speed motor which then controls the lime feed accordingly.

The cyanide analyzer [2] consists essentially of two metering pumps, an oscillating silver indicator electrode, a reference electrode, and a potentiometer. The sample pump is set at 2 ml/min and requires a filtered solution. The analyzer is standardized by the reagent pump, using the prepared reagent solution and a known cyanide standard. Prior to this the recording apparatus is calibrated by several known standards in order to establish a satisfactory reference point.

The complete recording apparatus consists of

(1) MV/I Transmitter

(2) Strip chart recorder

(3) Basic controller

An electric signal from the controller operates a pneumatic valve to control the addition of cyanide solution and thus maintain the desired level of free cyanide.

It was originally assumed that cyanide and lime feeds controlled by an operator would, over the period of a day or so, produce a mean close to the desired level. (Fig. 3a). This assumption was proved to be incorrect when recordings indicated extreme variations for periods lasting several days (Fig. 3b).

With the feed apparatus on automatic control very close correlation to the desired level has so far been obtained. (Fig. 4).

Further data is required before a final report can be made on this particular process control system. To date, the indications are that lime consumption can be reduced by almost 50% and cyanide by 25% in addition to better metallurgical results. A significant improvement indeed.

I wish to thank the senior personnel of Bralorne Pioneer Mines Ltd. for permission to write this paper and to express my appreciation to this Convention for the privilege of presenting it.

(1) W.A. Gow, H.H. McCreedy, F.J. Kelly Extraction Metallurgy Division Internal Report EMA 65-3.

(2) "The Measurement of Free Cyanide Concentration by Continuous Potentiometric Titration", Ingles, J.C. Mines Branch Research Report T. 127 July 1964

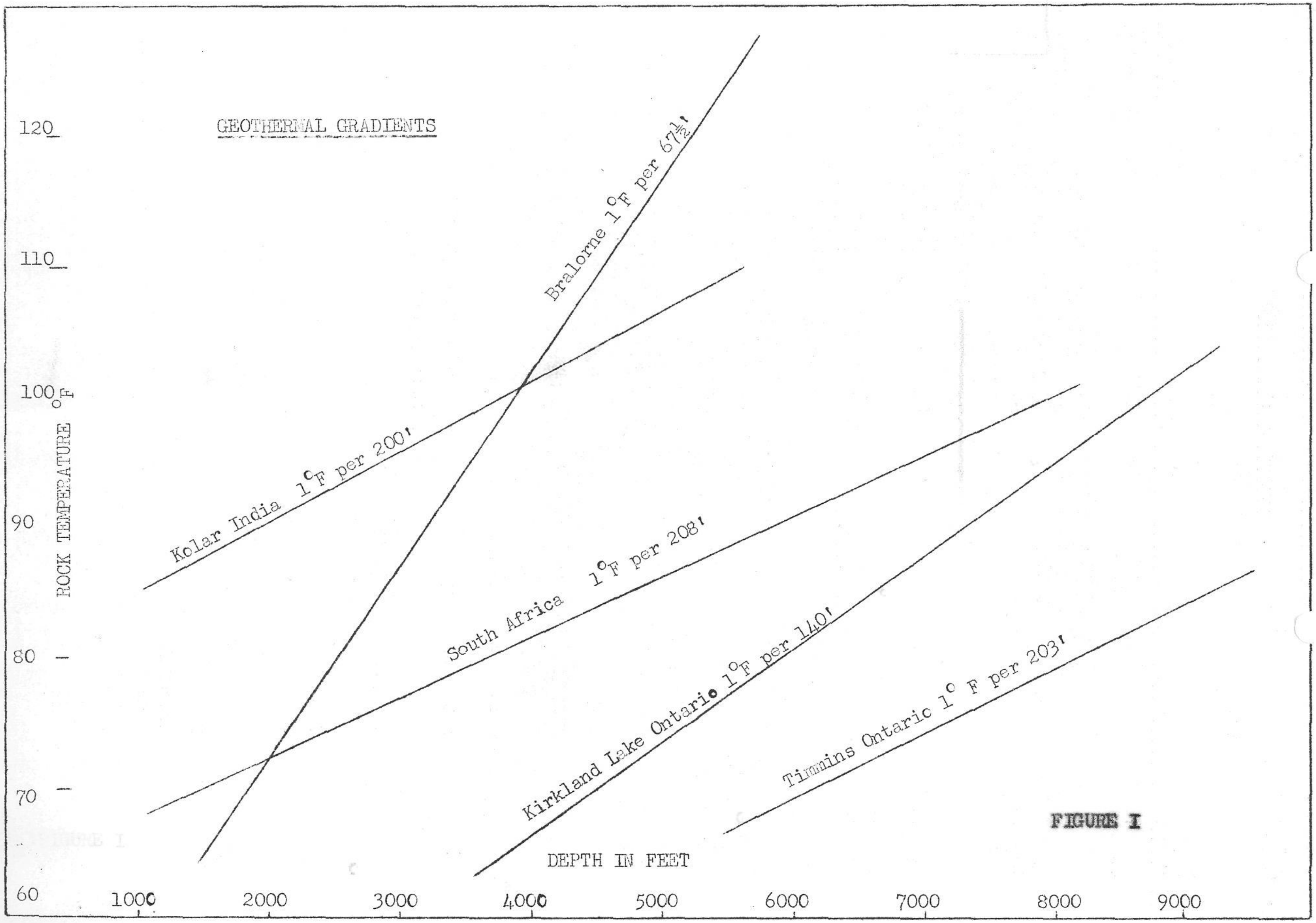

GEOTHERMAL GRADIENTS
ROCK TEMPERATURE °F
120
110
100
90
80
70
60
Bralorne 1°F per 67½'
Kolar India 1°F per 200'
South Africa 1°F per 208'
Kirkland Lake Ontario 1°F per 140'
Timmins Ontario 1° F per 203'
DEPTH IN FEET
1000
2000
3000
4000
5000
6000
7000
8000
9000

FIGURE I

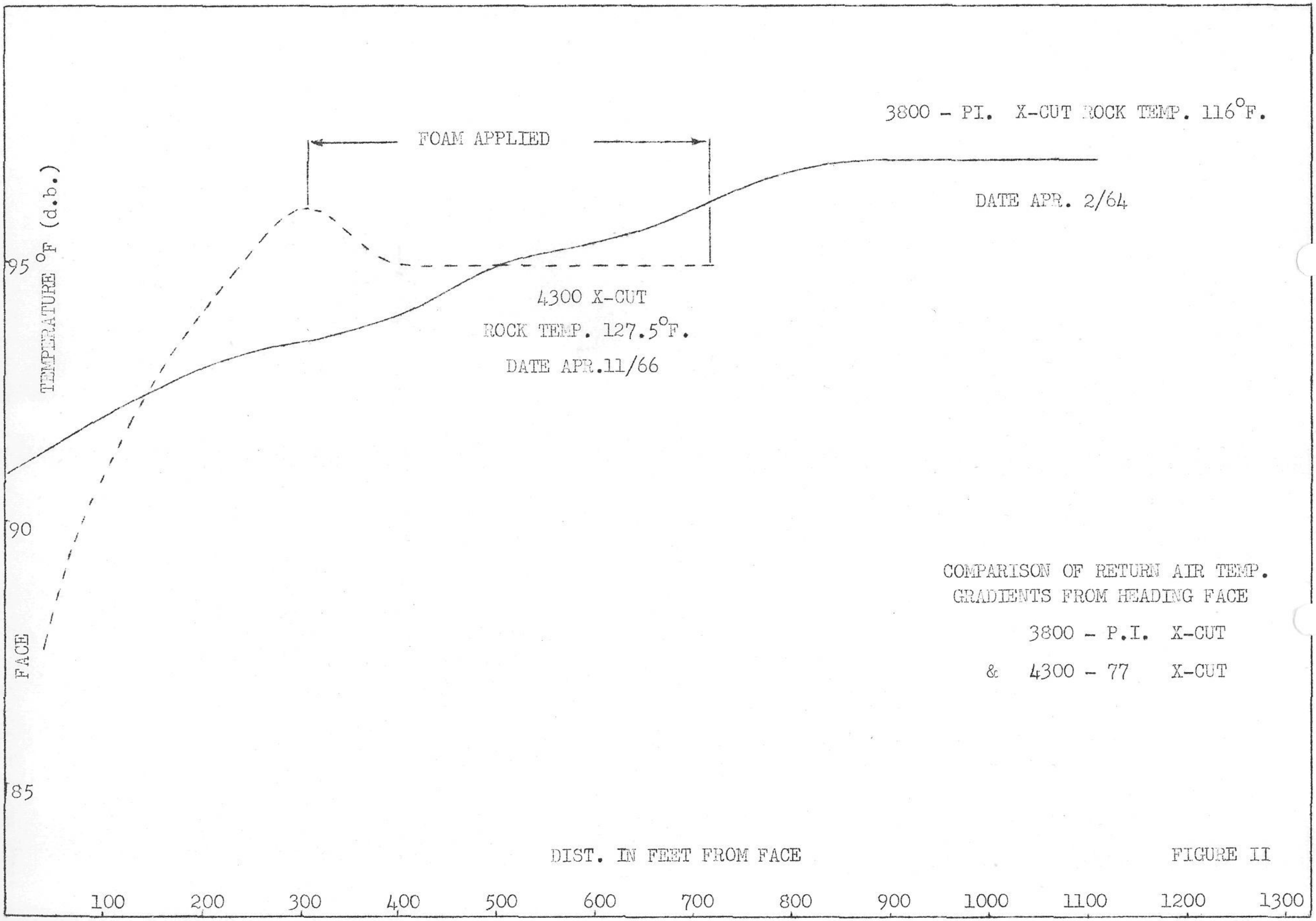
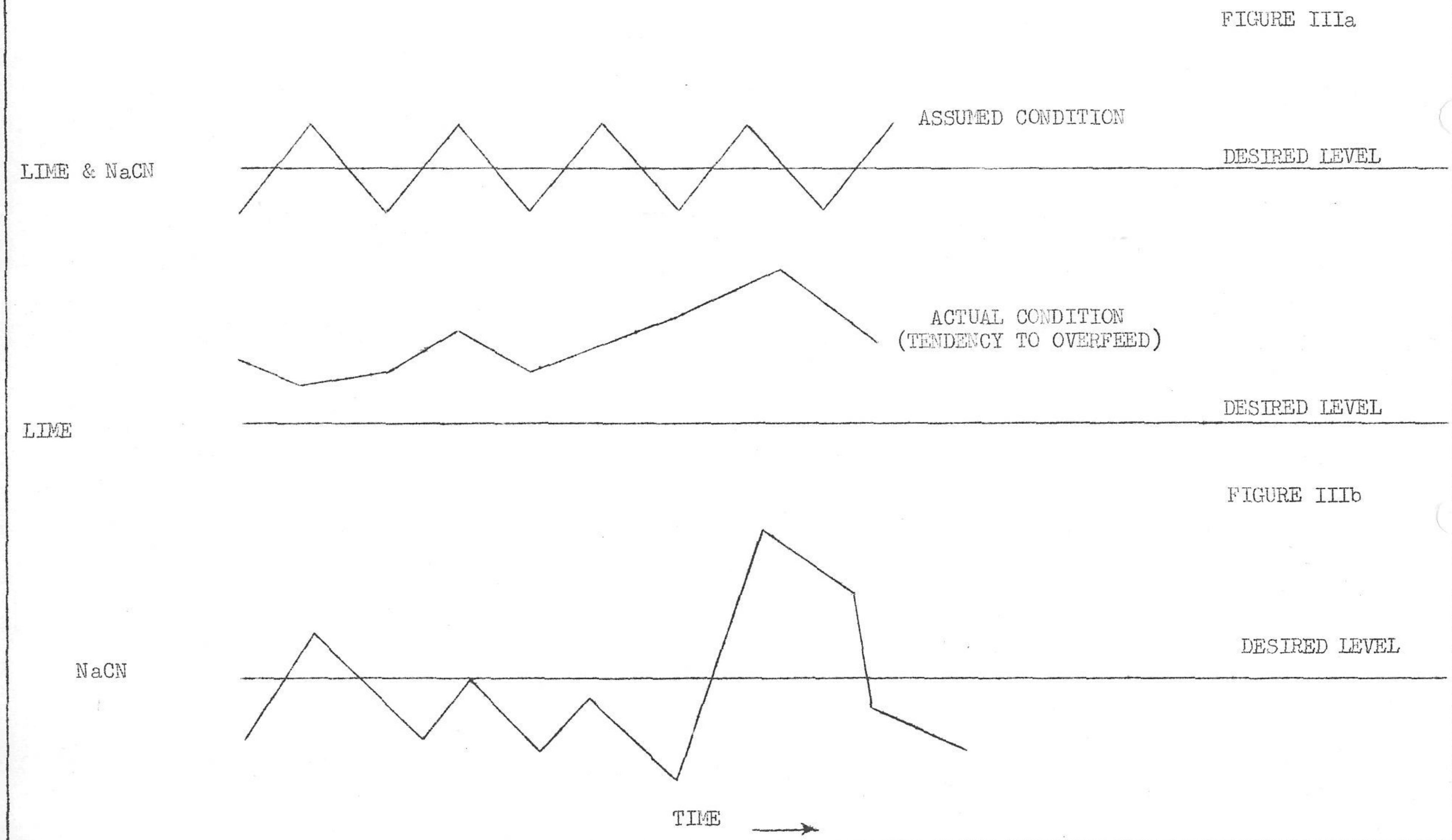

3800 - PI. X-CUT ROCK TEMP. 116°F.
FOAM APPLIED
DATE APR. 2/64
TEMPERATURE °F (d.b.)
95
4300 X-CUT
ROCK TEMP. 127.5°F.
DATE APR.11/66
90
COMPARISON OF RETURN AIR TEMP.
GRADIENTS FROM HEADING FACE
3800 - P.I. X-CUT
& 4300 - 77 X-CUT
FACE
85
DIST. IN FEET FROM FACE
100
200
300
400
500
600
700
800
900
1000
1100
1200
1300

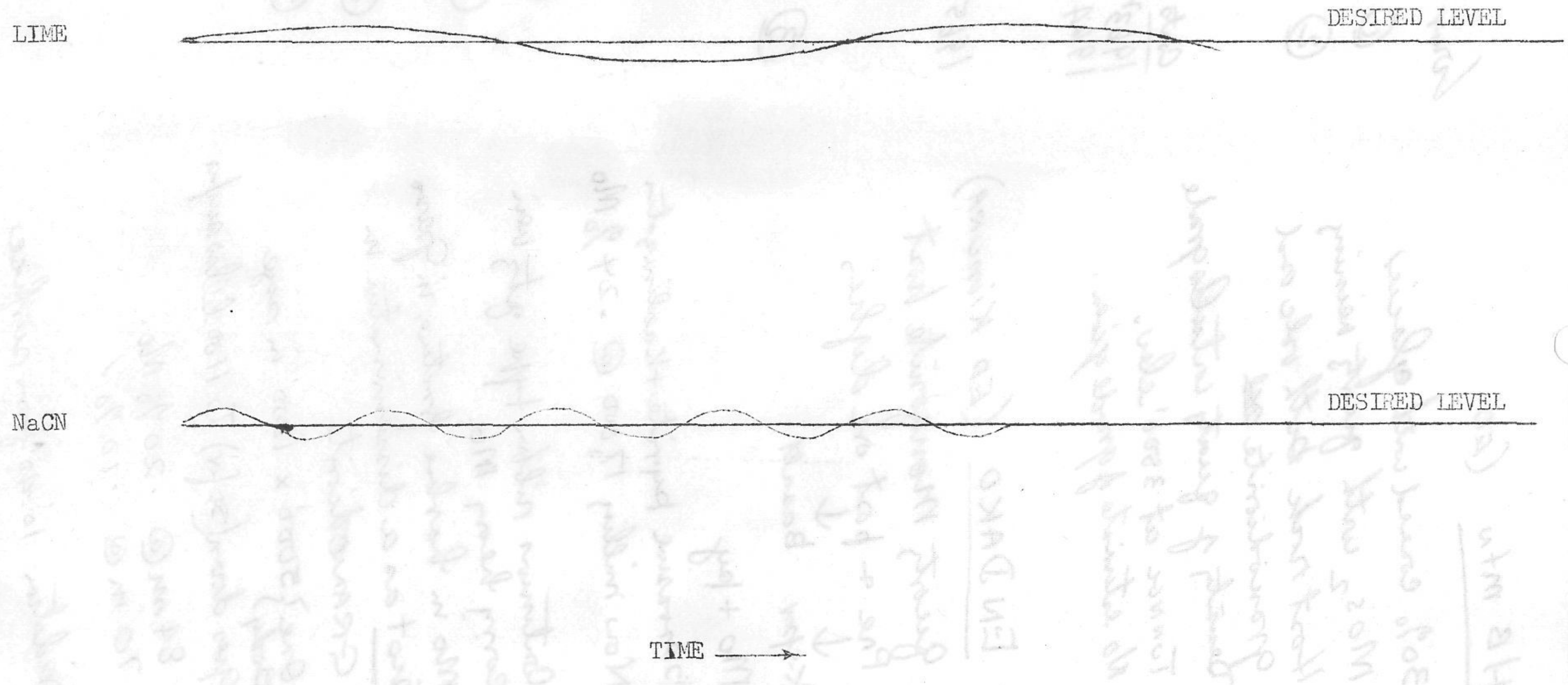
FIGURE II

GRAPHS SHOWING TYPICAL VARIATION IN REAGENT CONCENTRATION
OPERATOR CONTROLLED FEED



GRAPHS SHOWING TYPICAL VARIATION IN REAGENT CONCENTRATION CONTROL
BY FEED APPARATUS

FIGURE IV



HB mtn (ED

1. 30% covered with glacier
2. MoS_2 with quartz veining
3. Host rock both volc and granodiorite ~~and~~
4. Density of quartz controls grade
5. Tonnage at 3500' elev.
6. No estimate of grade given.

ENDAKO (ED KIMURA)

1. Quartz Monzonite host
2. Pre + post ore dykes
   ↓ K-spar ↓ Basalt
3. Mo + py
   Pervasive pyrite + Kaolinization
4. Now milling 17,000 @ .24% Mo
5. Continuous ribbon type qtz vein carry heavy Mo.
6. Mo in hairline fractures in Grano not as a dissemination in GRANODIORITE
7. One Body } 5000' x 1200' in size
   Goes down (so far) To 1100 below surface
   84 m @ .20% Mo.
   + 70 m @ .10% ✓
8. Oxidation 10'-40' from surface
   NX - NQ wireline used
   poor recovery ✓

S.IT CK: (R. McCrae

Copper Deposit
Lower STIKINE AREA

1. 2000 - 4500 A.S.L.
2. 7000' x 3000' pyrite halo in Volcanics - ANDESITES
   py + cupy

Costs
1963 men 6 - $17,000 for 36 days } No d.d.
1964 ✓ 16 - 69,000 ✓ 60 ✓
Aver cost/MAN - $25/man/day

1965 dd 7200 (BQ) $208,000
Overall cost per foot - $25/ft.

3. Pictures suggest steep topo like Lucky LAKE

GRANDUC (R. GUTHRATH)

1. Exploration of area in 1955-59
2. EACH step predicated on the previous step.
3. dithizone field kit used - THM
4. Host rock - Feldspar porphyry Kaolinized.
   - Cupy best in qtz-feld zone 300 wide.
   - Cu + Mo + py carrying qtz