

## Agnew Lake Mines: Taking giant steps in solution mining

PIONEERING the commercial application of in-situ leaching and surface heap leaching to Canadian uranium ore, Agnew Lake Mines Ltd. is working hard to get the kinks out of an operation designed to produce 1 million lb per year of  $U_3O_8$  concentrate. The Agnew Lake project, 37 mi west-northwest of Sudbury, Ont., is the sixth and newest uranium producer in Canada. After two years of pilot testing, the decision to go into full-scale production was made in March 1976.

Agnew Lake is owned 90% by Kerr Addison Mines Ltd. and 10% by Uranerz Exploration and Mining Ltd. In its latest annual report, Kerr Addison stated that mine development and leaching problems, together with overly optimistic performance expectations, "have resulted in the quantities of ore being broken in stopes being considerably below plan, and have prevented leaching underground on a production scale" (E/MJ, July, p 43). When E/MJ visited the property in July, delivery commitments were being met by borrowing from Eldorado Nuclear Ltd. and by heap-leaching ore on the surface.

The pioneer operation uses heavy blasting of ore to achieve extreme fragmentation (minus 6 in.), followed by percolation of dilute sulphuric acid through the in-situ broken ore and collection of the percolated solution, with dissolved uranium values, at a sump level. In addition to in-situ leaching, blasting "swell," which amounts to about 30% of the ore volume, is brought to the surface for heap leaching.

While difficulties caused by complex local faulting can be solved by additional development, the leaching problems, caused by inadequate distribution of leach liquor to the stopes, require a change in design from "trickle" spray distribution to a "flood leaching" mechanism. At last report, concrete bulkheads were being constructed on solution-collection levels to accommodate the new flooding concept.

Initial development of the mine took place during 1968-70, when Kerr Addison was planning a conventional underground operation. The production shaft was sunk to 3,400 ft and crosscuts were completed on six levels. However, low uranium prices forced a temporary closing of the mine in 1971. The mine was brought out of care-and-maintenance status in 1974 when the uranium price increased, and a two-year pilot operation began, testing the feasibility of in-situ leaching in an experimental 50,000-ton stope (E/MJ, January 1976, p 100).

The deposit being mined is a highly fractured oligomictic quartz pebble conglomerate containing uranothorite as a matrix constituent. It occurs in two known multiple-bed zones—the "3 Zone" and the lower "5 Zone"—that are stratigraphically separated by about 600 ft. The beds strike east-west at the surface, dip 65-85° south, and average 17.2 ft in thickness. The 3 Zone has been traced to a depth of 3,100 ft, and has undergone major faulting, as has the 5 Zone. Movement along east-west-trending faults has been both horizontal and vertical, resulting in a south-side-down displacement



Stoping is done with downhole drills that create 4½- and 6-in.-dia holes. Average hole depth is 100 ft.

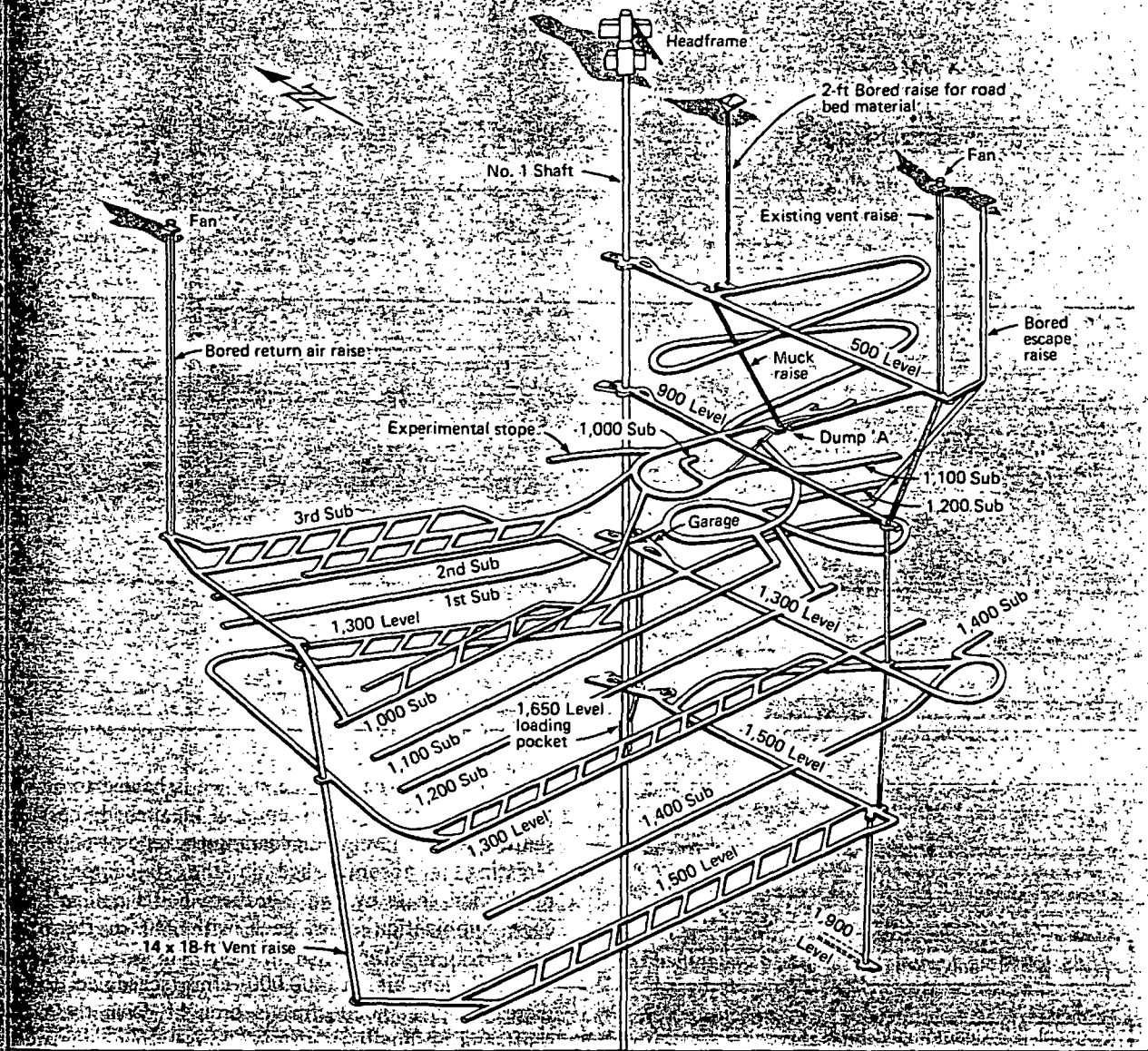
and creating a fairly complex geologic picture. The parent rock is Huronian arkose lying unconformably on Prehuronian granite.

The deposit was discovered in the early 1950s by ground prospecting and was initially drilled as a thorium property. It has an average grade of 1.0 lb  $U_3O_8$  per ton of ore. Proven and probable reserves amount to 12 million tons.

When in full production, the mine will employ 350 workers underground and 150 on the surface to produce 4,800 tpd. The mine is currently developed on eight main levels, 100 ft apart. On each level, pilot headings 18 x 12 ft high are driven along strike from the production shaft and are slashed to the width of the ore. Hangingwall extraction drifts (18 x 10 ft) are then driven at vertical intervals of 200 ft, from which drawpoints to the ore are located at 65-ft horizontal intervals. Diamond drilling from these drifts provides a detailed outline for stope drilling layouts. Stope blocks have averaged 1,200 x 250 x 17.2 ft.

Most development work in the mine is done with two-

Isometric view of underground mine workings, Agnew Lake Mines



and three-boom electric-hydraulic jumbos equipped with Montabert rock drills. Stopping is performed with 10 Ingersoll-Rand downhole drilling machines. Development ore and swell from blasting are hauled by Wagner ST 8A and 5A "Scooptrams." Heavy equipment is deployed by using the mine's spiraling service ramp.

Stopping with downhole drilling equipment creates 4½-in. blastholes spaced 6 ft apart, with 7-ft burdens and average depths of 100 ft. The stopping technique is similar to that used in Inco's Sudbury nickel mines—with slot raises drilled at one extreme of a stope block to connect an upper sublevel and a stope undercut. Ore is blasted in slabs, retreating from the original free face.

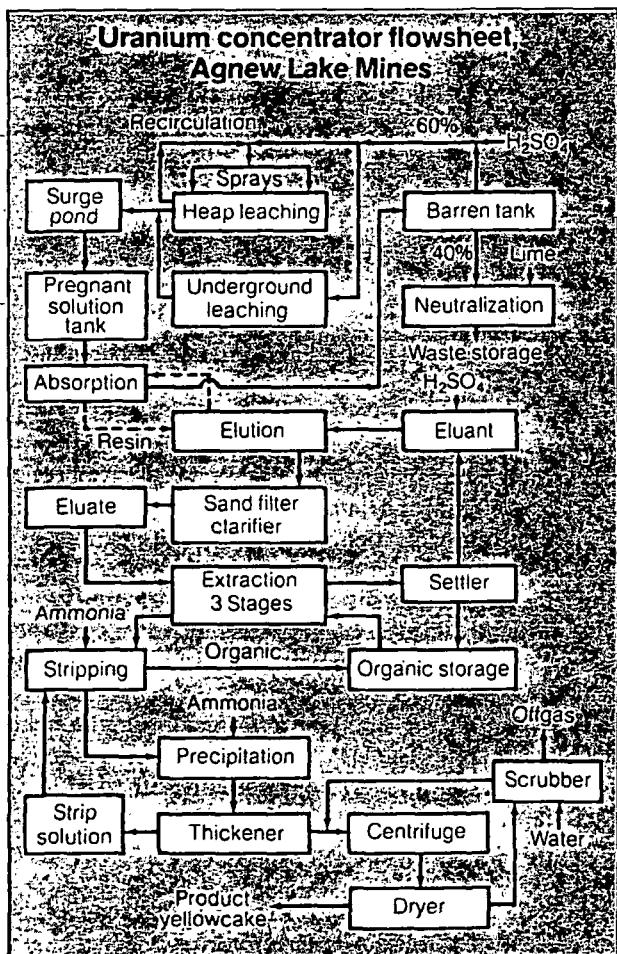
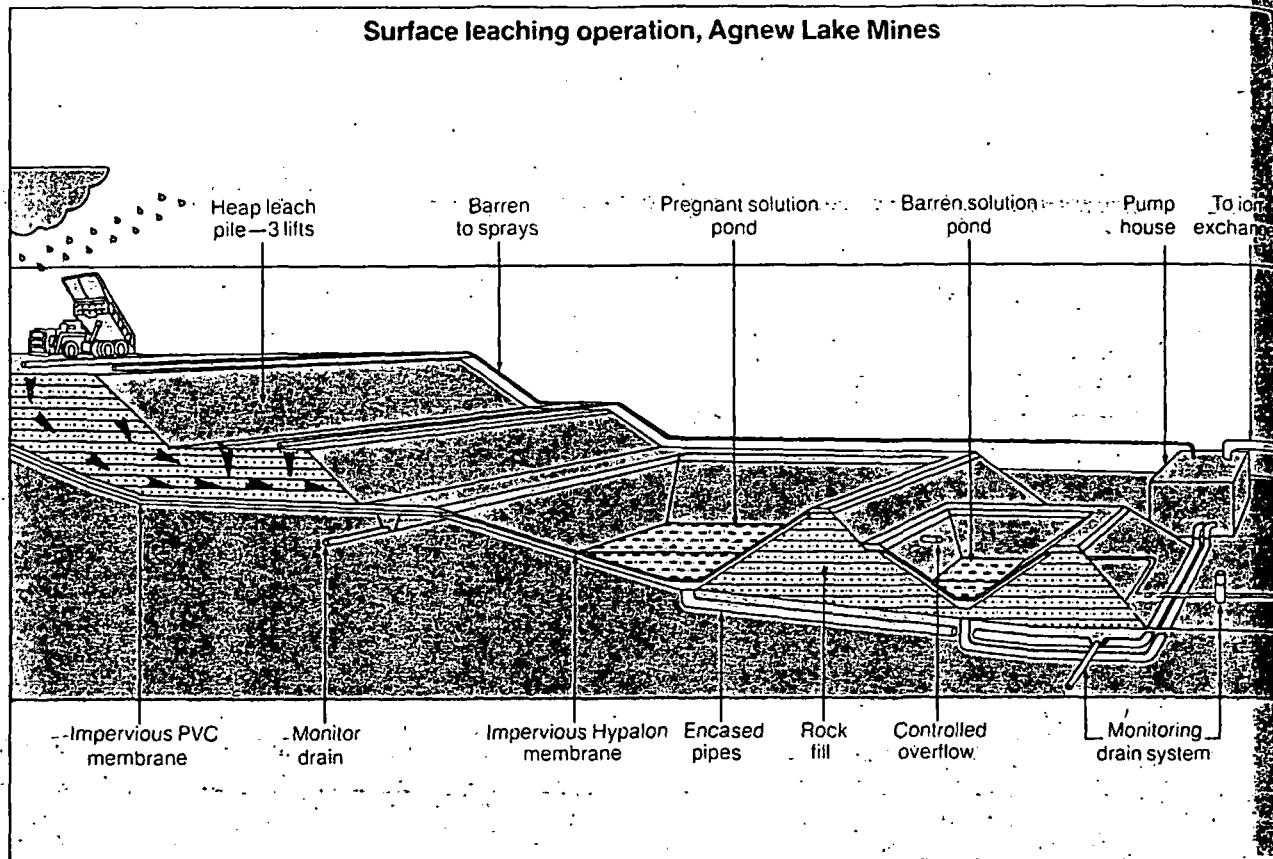
Stope blasting is done with "Anfomet" or "Hydro-mex" at a powder factor of 1.8 lb of explosive per ton of rock. In each round, roughly 2,000 tons are reduced to minus 6-in. fragments, and the swell is pulled off through extraction drifts.

In-situ leaching is performed on 100- and 200-ft "lifts." During E/MJ's visit, Agnew Lake was leaching

levels 1,300 through 1,500 using a "trickle" technique, in which 4-in.-dia ABS perforated pipe is run along the top of a broken ore stope, and sulphuric acid at pH 1.5 is pumped through. The acid percolates through micro-cracks in the ore and induced fractures and is collected on the 1,500 sill and pumped to the surface. At this point,  $U_3O_8$  concentration is about 0.25-0.3 gpl. Because the steeply dipping nature of the orebody has created problems in solution distribution, stopes in the future will be literally flooded with leach liquor, after construction of concrete plugs on sump levels to contain the acid.

Only 30% of the uranium is originally in the acid-soluble  $U^{+6}$  state. The  $U^{+4}$  that makes up the remaining 70% must be converted to  $U^{+6}$  by seeding stopes with the bacterium *Thiobacillus ferrooxidans*. The bacteria assist in the conversion of pyrrhotite and pyrite found in the waste rock from  $Fe^{+2}$  to  $Fe^{+3}$  state, and the ferric ions oxidize  $U^{+4}$  to soluble  $U^{+6}$ . During leaching, some of the ferric iron is reduced to ferrous iron.

Surface leaching operation, Agnew Lake Mines



Ore to be leached on the surface is hauled to an on-pass running through the center of the orebody near the shaft, from 900 level to the loading pocket on 1,630 level. Hoisting is accomplished in the six-compartment production shaft by two Nordberg hoists pulling skip-cage combination units, each with 240 cu ft of ore space.

Ventilation air at 400,000 cfm (scheduled to be increased to 800,000 cfm) is brought down the main shaft and an auxiliary airway, flows through the work areas, and is exhausted through the east and west extremities of the mine. A Subterranean raise bore capable of reaming 10-ft raises is used to create ventilation openings to the surface; it is described by mine engineers as "an excellent investment."

Ground support is provided by 6- and 8-ft rock bolts. In addition, 4-in. wire mesh screen is used in bad ground. Water is pumped to the surface at the rate of 225,000 gpd using two Smart-Turner multistage pumps, one on 1,900 level and one on 900 level.

Heap leaching on the surface

The stockpile leaching area is a natural depression that has been contour-graded for positive drainage of leach solutions to a pregnant solution pond (see surface leaching diagram). The area, covering 600,000 sq ft, is lined with a 20-mil PVC cover, on top of which lies an 18-in. layer of sand that protects the PVC from damage. Below the membrane, a French drainage system has been installed to monitor possible leaks.

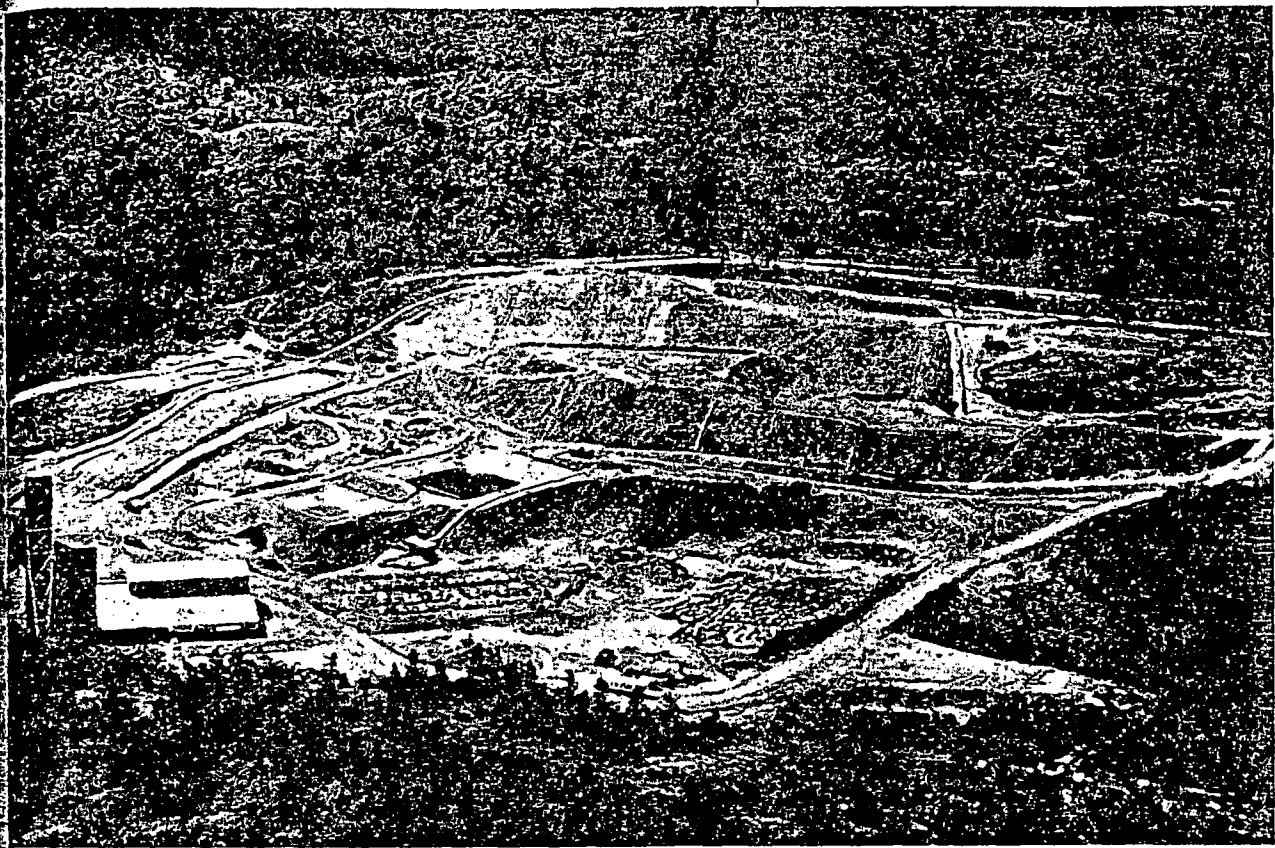
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Leach dumps containing 1.5 million tons of ore (center) are the dominant feature of the Agnew Lake surface operation.

When E/MJ was on site, the surface dumps contained 1.5 million tons of ore overlain by about 30,000 ft of perforated 2- and 6-in. pipe bearing the leach solution.

At that time, the ratio of ore being leached on surface to that leached underground was roughly 3:2. Agnew Lake has been contracting for use of a portable crushing plant that reduces ore to minus 3 in. before it is dumped onto a leaching heap.

Leach solution that has percolated through the heap drains into a Hypalon-lined 1.6-million-gal pregnant solution pond. From there, it is pumped to the ion exchange processing plant. A smaller adjacent pond, also Hypalon-lined, stores barren leach solution and acts as an overflow basin for the pregnant pond in case of emergency.

**Plant features Himsley continuous contactors**

Since mill feed is in the form of solution, the mill needs no crushing or grinding circuits. Solution undergoes two stages of purification by means of the USAEC-developed Eluex process, a combination of resin ion exchange and solvent extraction (see flowsheet). Resin ion exchange is done in two parallel sets of Himsley moving-bed ion exchange columns in which pregnant solution flows upward via the resin. These continuous fluid-solid contactors use standard ion exchange resins in substantially lesser volumes than fixed-bed units. They can treat turbid liquids, and they operate automatically using several fail-safe mechanisms. Agnew Lake is the first commercial installation to use the continuous

contactors.

Pregnant liquor from storage is pumped through the Himsley units, where uranium is adsorbed onto "Duolite A 101-D" strong-base anionic resin. As the resin becomes loaded, it is automatically moved to elution columns, where uranium is stripped from the resin by backwashing with strong sulphuric acid. The eluate is then passed through a sand clarifier before it is introduced into the solvent extraction circuit.

Barren solutions from the Himsley columns are split—approximately 60% going to the leach circuits after additions of sulphuric acid, and 40% to waste disposal after lime neutralization and barium chloride treatment. The waste solution is then pumped 7,000 ft through pipes-within-pipes to a 25-acre tailings area. The barium chloride is added to precipitate small amounts of radium<sub>226</sub>. No water is recycled to the plant from the tailings pond.

In the solvent extraction circuit, the eluate is stripped in three stages using an organic that consists of a tertiary amine in a kerosene carrier. The resulting uranium-rich amine is then contacted with an ammonium sulphate solution that strips the organic of uranium and with ammonia to raise the pH to 4.5. The pregnant solution is treated with more ammonia to precipitate the uranium as ammonium diuranate.

The precipitated ammonium diuranate yellowcake is thickened and centrifuged, then dried in an Eimco hearth-type dryer and packed in 45-gal drums for shipment. The grade of the final product is equivalent to 88-91% U<sub>3</sub>O<sub>8</sub>. □

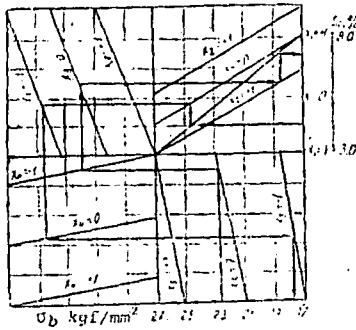


Fig. 1 Nomogram for the dependence of the ultimate strength on the chemical composition and thermal-time parameters of the heat treatment of the charge. Forward code: Si-Cu-Mg-t<sup>0</sup>-T. Reverse code: T-t<sup>0</sup>-Mg-Cu-Si.

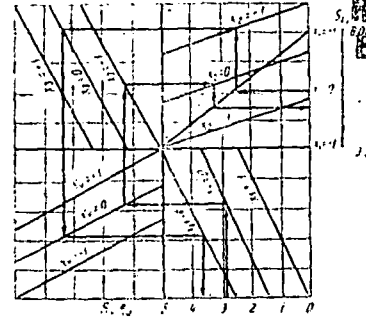


Fig. 2 The nomogram for the dependence of the relative elongation on the chemical composition and thermal-time parameters of the heat treatment of the charge. Forward code: Si-Cu-Mg-t<sup>0</sup>-T. Reverse code: T-t<sup>0</sup>-Mg-Cu-Si.

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treatment conditions for the charge of the alloy AL14Ch containing 6% Si, 3% Cu, and 0.5% Mg. Here the condition was imposed that the ultimate strength should not be less than 24kgf/mm<sup>2</sup> after casting in a dry sand mould. The desired charge heating temperature was 500°C, and the holding time was 3.5h. This gave a strength of 24.6 kgf/mm<sup>2</sup>. The same alloy prepared without previous heat treatment of the charge has a strength of 18.3kgf/mm<sup>2</sup>.

It is thus possible by means of the obtained mathematical relationships, expressed in the form of nomograms, to select the optimum parameters for the heat treatment before the melting of secondary aluminium alloys, supplied to the casting plant from metallurgical works, with a view to the production of a high level of strength and ductility in the castings.

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References

- 1) V F Shleptsov et alia in Melting and casting of non-ferrous metals and alloys: Metallurgiya, Moscow 1969, p. 112.
- 2) F N Strel'tsov and Yu M Leibov: Tsvetnye Metally 1973, (9), 45.
- 3) Yu M Leibov et alia in Improvement of the production technology for intermediates of non-ferrous metals: Moscow 1969, p. 41.
- 4) G G Kruchenko et alia: Byull. TsIIN. Tsvetnaya Metallurgiya 1974, (20), 34.
- 5) S I Torshilova et alia: Izv VUZ Tsvetnaya Metallurgiya 1976, (4), 126.
- 6) I F Kolobnev: Heat treatment of aluminium alloys: Metallurgiya, Moscow 1966, p. 394.

UDC 669.2

Automatic optimisation of some processes in concentration and the hydrometallurgy of non-ferrous metals

G G Rannev (Krasnodar Polytechnic Institute - Department of Automation)

In papers by Eykhöff<sup>1)</sup> and Gill<sup>2)</sup> it was shown that with clear parabolic characteristics in the subject the search with it cannot in principle be eliminated or replaced by a search in a model of the subject. The question as to why trial steps on the subject or 'teeth' are required for the characteristics of the open part, irrespective of whether or not a model is used in the external system, was examined in detail in the literature<sup>3)</sup>.

The majority of processes in the concentration and hydrometallurgy of non-ferrous metals are subjects with a clear and often approximated (with sufficient accuracy for practical purposes) parabolic characteristics. Such systems, therefore, require trial steps on the subject in principle, and all the numerous attempts to create non-search systems with these subjects are hopeless and erroneous.

A system of automatic optimisation of such processes must clearly have open, search and correcting parts, which have the following functions:

1. The open part realises the automatic control of the process in the period between the searches according to the results from the last search (or in the initial period of control based on the data from technological instruction);
2. The search part realises determines the coefficient of the open part by means of continuous or brief disturbances introduced at the head of the process (without causing substantial fluctuations) and by recording the current values of the optimised parameters at given points. The optimum regime is determined by one of the known methods, or, better, by the random search method<sup>4)</sup>, which has an advantage over other methods with the large number of controlling influences usually encountered in the processes investigated.

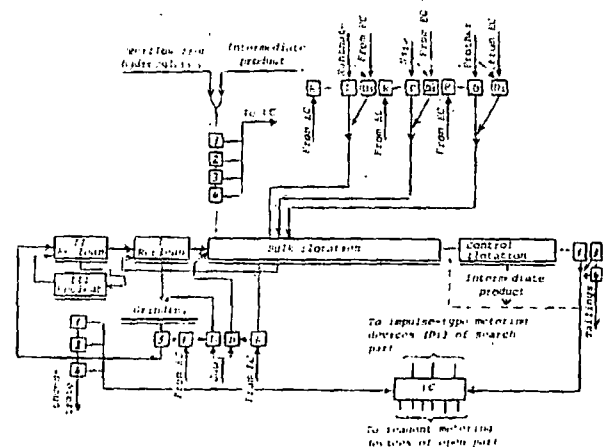


Fig. 1 The system for automatic optimisation of the flotation of mixed copper ores. 1) Induction-type flowmeter; 2) densimeter; 3) concentration meter for the oxidised copper in the pulp; EC is an electronic computer; D is a reagent metering device; R is a regulator; 5) concentration meter for lime.

The corrector serves to memorise the values of the optimum coefficients, to calculate the current values of the controlling influences from them, and to correct the characteristics of the open part on their basis. We developed optimisation systems, block diagrams of which are given in figs. 1 and 2, as applied to the flotation and electro-deposition of zinc<sup>5)</sup>. The operation of the systems was ex-

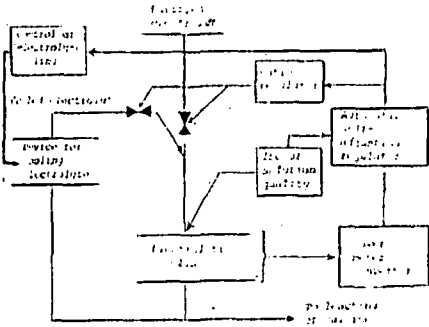


Fig. 2 The system for automatic optimisation of the electrodeposition of zinc. ICS = information control-optimum-graphic system.

ained earlier<sup>5)</sup>), and the basic definitions are given in the figures.

The system for automatic optimisation of the flotation process realises the task of obtaining the maximum extraction of the valuable component and the minimum consumptions of the reagents for a given quality of concentrate and a given quality in the initial feed. Its algorithm is as follows:

$$\epsilon^* = F(Q_p, \beta, \alpha) \rightarrow \text{max. possible}$$

$$Q_p < Q_{\min}$$

$$\beta \geq \alpha, \alpha = \text{var.}$$

where:

- $\epsilon^*$  = the extraction %;
- $Q_p$  = the consumption of the reagents g. ton (g/kg);
- $\beta$  = the content of the metal in the concentrate %;
- $\alpha$  = the content of the metal in the ore %;
- $a$  = the given (planned) quality of the concentrate %.

The system for automatic optimisation of the electrolysis of zinc realises the task of producing the maximum possible current yield of zinc with the minimum possible electricity consumption for a given quality in the solutions delivered for electrolysis. Its algorithm is as follows:

$$Q = Q_1(x_1, x_2, \dots, x_n)$$

where:

- $x_1$  = the temperature °C;

- $x_2$  = the acidity g. l;
- $x_n$  = the concentration of impurities and other technological factors.

A mathematical indication of the effectiveness of the zinc electrodeposition process is the equality of the partial derivatives to zero at the extremum<sup>6)</sup>:

$$\frac{\partial Q_i}{\partial x_i} = 0 \quad (i = 1, 2, \dots, n)$$

Technical and economic calculations on the use of the systems described above and the results from the normal operation of the investigated processes showed that the use of the automatic optimisation systems will lead to a saving of 1 136 000 roubles a year at the Almayk copper concentration plant, 208 000 roubles at the Almayk zinc plant, and 430 000 roubles at the Chelyabinsk electrolytic zinc works. The results from introduction of the system of automatic optimisation of the hydrometallurgical zinc production process at the Almayk electrolytic zinc works confirmed the correctness of the approach to the solution of problems of this type<sup>7)</sup>.

All this confirms that the automatic optimisation of concentration and hydrometallurgical processes with clear characteristics in the subject must only be realised by means of search systems.

#### References

- 1) P Eykhoff: IRE Trans. Vol. Ac-5, 1960 (June) No. 2.
- 2) A Gill: Introduction to the theory of thermal automation devices: Nauka, Moscow 1966.
- 3) A G Ivakhnenko: Self-teaching systems of recognition and automatic control: Tekhnika, Kiev 1962, p. 28.
- 4) L A Rastrigin: Random search: Izd. Akas. Nauk LatvSSSR 1964.
- 5) G G Rannev et alia: Author's Certificate No. 367 172 Class C22 1/22 with priority from 14.09.64. Bulletin No. 8, 1973.
- 6) G G Rannev et alia in Automatic control and regulation in the concentration and hydrometallurgy of non-ferrous metals: Tashkent 1972, p. 119.
- 7) G G Rannev et alia: Reports of All-Union Scientific-Technical Conference on "The state of the development and introduction of automated systems for the control of technological processes in non-ferrous metallurgy: Moscow 1975, p. 74.

AUTOCLAVE PRECIPITATION OF COPPER FROM MADNEULI COMBINE CHALCOPYRITE  
CONCENTRATE LEACHING SOLUTIONS

UDC 669.334.92: 66.046.8

S. S. Naboichenko, V. P. Bogdashev, V. K. Pinigin, T. N. Zashikhina  
and T. P. Geleishvili

In collaboration with the Nickel Industry Research and Design Institute the Institute of Physical and Organic Chemistry, Academy of Sciences of the Georgian SSR has proposed an autoclave oxidizing technology (170-180°C, excess oxygen pressure, optimum acidity buffering) for leaching chalcopyrite concentrates from the Madneuli Mining and Concentration Combine; in this process up to 96 - 99% Cu is extracted into solution selectively from iron. Autoclave precipitation with hydrogen is one of the methods of extracting copper from the solutions produced.

The present work gives the results of laboratory and larger-scale tests to produce powder from solutions containing, g/liter: 40.7-42.0 Cu, 3.5-6.7 Fe (including 3.0-5.6 Fe<sup>2+</sup>), 33-40 Mn, 7.8-8.5 Zn, 4.0-6.7 Mg, 7.8-13.2 H<sub>2</sub>SO<sub>4</sub>, < 0.1 Ca<sup>2+</sup>, Ni<sup>2+</sup>, Al<sup>3+</sup> Si<sup>4+</sup> (each); traces of Sb and Co.

In the first experiments there were substantial deposits of powder on the inner surface of the autoclave at a polyacrylamide (PAA) consumption of 0.01 g/g. Attempts to reduce deposition by reducing the temperature and the hydrogen pressure and by seeding did not give positive results. Only an increase in the polyacrylamide consumption to 0.015-0.2 g/g Cu reduced deposition to 4-8%; in these circumstances, however, a loose copper deposit in sponge form was produced, due to formation of conglomerates with a highly developed powder particle structure. With a temperature of 150° and a PAA consumption of 0.015-0.02 g/g Cu the fineness of the powder increased and the particles became more rounded in shape, which improved the settling and de-watering of the deposit. The use of the preparation PuMA at the rate of 0.005-0.007 g/g Cu proved to be more effective; this reduced the deposition of copper on the walls to 3-4% and also substantially reduced the unit consumption of additives.

As a rule the fineness of the powder increases when the PAA consumption increases; however, by virtue of subsequent grinding this difference is not important, and the average powder particle size is 30-40 μ. It is interesting to observe that the powder consolidates at a PAA consumption of 0.015-0.02 g/g Cu and a temperature of 150°C.

It was established that the following conditions were essential to produce a residual copper content of not more than 2.0 g/liter\*:  $t > 140^{\circ}\text{C}$ ,  $P_{\text{H}_2} > 25 \text{ at}$ , and  $\tau > 30-40 \text{ min}$ .

Experiments in processing model solutions containing (in g/liter) 40 Mn and 62 and 81.7 Cu showed that the average speed of deposition increased from 1.0 to 1.45 and 1.75 g/(liter.min) with an increase in the solution initial copper content; however, the deposit was spongy and the residual copper content reached 6-12 g/liter ( $t + 140^{\circ}\text{C}$ ,  $P_{\text{H}_2} = 28 \text{ at}$ ,  $\tau = 40 \text{ min}$ .)

Kinetic experiments in a 25-liter autoclave to assess the effect of process parameters upon the process results and the properties of the produced powders showed (Table 1) that a temperature of 140-150°C and  $P_{\text{H}_2} = 28-30 \text{ atm}$  were required to precipitate no less than 97% Cu at an average speed of  $v = 1.1-1.3 \text{ g}/(\text{liter}\cdot\text{min})$ . The bulk density  $\gamma$  of the powders increased and their specific surfaces decreases with a rise in the hydrogen pressure and the temperature. However, the effect of these parameters is insignificant: thus the bulk density, fineness, and specific surface altered by 15-20% on average when the hydrogen pressure increased from 15 to 30 at or the temperature rose from 130 to 160°C.

Grinding the powder ( $\tau = 10 \text{ min}$ , rod mass 1 kg) leads to an increase in bulk density and a reduction in the specific surface (see Table 1, data in parentheses); this is more marked in powders produced at elevated temperatures (by 40-60% on average) than in powders produced with variation in the hydrogen pressure (by 10-15%). Apparently the temperature is a decisive factor in forming the powder particle structure.

Data on settling ( $t = 20^{\circ}\text{C}$ , initial solid-to-liquid ratio 20) indicate a high separation speed (see Table 1). Routines for pulp dewatering and for washing and

\* Taking into account the requirements of further solution processing to obtain electrochemical manganese dioxide.

stabilizing the powder with naphthenate soap solution were studied in a screen-bowl centrifuge (rotor diameter 200 mm). Separation factor  $F$ , Pulp temperature and duration have the greatest effect upon dewatering. A powder residual moisture content of 10-15% is achieved with an  $F$  of 200-300, at 40-60°C, after 10-15 min processing. Hydrophilic additions at the rate of up to 0.1% of powder mass reduce its residual moisture content (other things being equal) by 4-8% (abs.).

Up to 80-90% sulfate sulfur is removed when the powder is washed with preheated artesian water (4-5 liters/kg, 40-50°C).

Precipitation of copper from a batch of actual solution 8.5 m<sup>3</sup> in volume was carried out<sup>1</sup> in a titanium autoclave battery (four vertical sections of 25 liters each) designed by the Nickel Industry Research and Design Institute; a 63-liter autoclave designed by the All-Union Chemical Machine Building Research and Design Institute and made from steel EI-943 was used for some of the experiments. The residual copper content did not exceed 1.0 g/liter after 30-35 min at 145± 2°C and  $P_{H_2} = 28 \pm 2$  at. With the

$P_{H_2}$  consumption adopted (0.7±0.1 g/liter) the powder deposits were slight; and even lower level of copper deposition on the inner surfaces was achieved in experiments with a PuMA consumption

of 0.2-0.3 g/liter. Loose accumulations of powder which readily broke up were observed in the stagnation zones in the autoclaves (on the diffuser upper glides, near the end wall, and above the heat exchangers).

The pulp was discharged from the autoclave at 50-70°C and dewatered in a pressure filter; the powder was washed with water at 40°C, then dried at 200-250°C. Some of the powder was accumulated and stored under a layer of mother liquor.

The latter contained, in g/liter: 0.3-0.8 Cu, 58-64 H<sub>2</sub>SO<sub>4</sub>, 31-33 Mn, 3.2-3.7 Fe, and 8.3-8.5 Zn and was suitable for further processing to obtain manganese dioxide.

Batches of dry powder and moist powder (12% H<sub>2</sub>O) of mass 250 and 100 kg respectively were processed in an industrial electric muffle furnace with a hydrogen atmosphere with the boats moving continuously at 28-30 mm/min. The dry powder was processed at 410±20°C with a hydrogen consumption rate of 16-18 m<sup>3</sup>/hr. Having regard to the gradual movement of the boats into the high-temperature zone and the overall time spent in the furnace (2-2.5 hr), the moist powder was processed like the first batch. The temperature of the powder on emerging from the cold zone was 25-30°C. The properties of the two batches of powder did not differ significantly, although direct pro-

Table 1  
Process Kinetics and Properties of Powders Produced with Various Deposition Routines (with addition of PAA 0.015 g/g Cu)

Test cond.,			Cu <sub>res</sub> g/l	Cu de- position, %	v, g/(l. .min) <sup>0.1</sup>	v <sub>res</sub> cm <sup>3</sup> /g	γ, g/cm <sup>3</sup>	s, m <sup>2</sup> /g.	δ, μm
t, °C	PH <sub>2</sub> , atm	τ, min							
150	30	20	0.32	98.1	2.90	—	0.52	1.87	82
150	20	30	0.69	98.3	1.84	9.52	0.49 (0.55) <sup>0.2</sup>	2.51 (2.37)	83 (60.9)
150	15	40	1.91	95.3	0.81 <sup>0.2</sup>	6.85	0.45 (0.55)	2.34 (2.22)	76 (55)
130	30	40	2.60	93.8	0.71 <sup>0.2</sup>	5.52	0.55 (0.78)	2.11 (1.4)	75 (49)
140	30	40	1.03	97.6	1.16	5.47	0.53 (0.71)	2.29 (1.35)	67 (63)
160	30	20	0.14	99.7	5.80	4.28	0.63 (0.84)	1.9 (1.33)	74 (56)
145	30	40	0.86	98.0	1.30	6.05	—	—	—

<sup>0.1</sup>By the 1.5 g/liter residual copper content

<sup>0.2</sup>Found by extrapolation

<sup>0.3</sup>Results after grinding powder in a mill are given in parentheses

Table 2  
Properties of Batches of Autoclave Powder Treated with Naphthenate Soap (I) and Hydrogen (II) --

Properties	I		II			
	Ordinary powder	Powder after grinding*	Dry powder	Wet powder	Combined powder	Powder after grinding*
γ, g/cm <sup>3</sup>	0.56	0.58	1.6	1.46	1.31	1.97
s, m <sup>2</sup> /g	4.37	0.081	1.18	1.43	1.74	0.052
Percentage of fractions (μm)						
100	8.0	2.9	25.5	15.8	7.1	8.2
74	27.0	7.3	13.8	9.0	18.4	11.0
44	26.4	50.3	13.7	17.8	23.2	37.3
-44	38.0	39.0	46.5	57.3	49.3	53.2
Content, %						
Cu		99.04			99.69	
Fe		0.004			0.004	
As		0.0003			0.0003	
S		0.003			0.002	
Sb		0.0007			0.0007	
residue insol- uble in HNO <sub>3</sub>		0.20			0.10	
O <sub>2</sub>		0.71			0.18	
Pb		0.004			0.003	

\* τ = 10 min

<sup>1</sup>V. F. Luk'yanchikov, S. M. Ushakov, V. I. Neustroev, and A. V. Kaigorodov took part in the work.



cessing of moist material yields a lighter powder with a highly developed surface.

The powder pulp was processed by a scheme for production of electrolytic powders<sup>2</sup> including centrifuging, washing, stabilizing with naphthenate soap in an FPD 1202 H-3 centrifuge ( $F = 500$ ,  $\tau \Sigma = 40$  min), drying the dewatered (9.1%  $H_2O$ ) powder in an electric boiler, and passage through a 100- $\mu$  mesh on a GZhD-4 vibrating screen. Stabilization of the powder was unsatisfactory, and there was increased dust generation on screening.

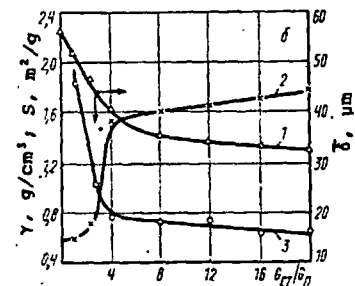
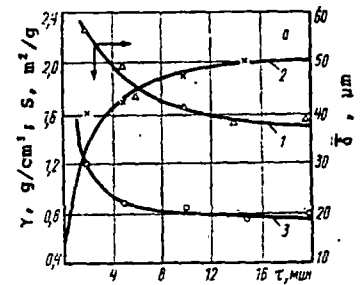
The properties of powders processed in a current of hydrogen and stabilized with naphthenate soap are given in Table 2. In the former case the powder is denser, has a lower specific surface value, and does not grind so well, but is purer than powder stabilized with naphthenate soap. The copper and insoluble residue content of the latter did not meet the requirements of State Standards for copper powder; however, the content of other impurities was several times lower. A special analysis of the "insoluble residue" showed that its principal constituents were rock-forming elements and carbon. This was due to the unsatisfactory clarification of the incoming solution and to excessively high consumption of organic additives. The loose structure of the powder particles obtained made washing and stabilization with naphthenate soap difficult and led to an increased carbon content.

To produce powder of the prescribed grade, the dried material is ground and screened within narrow fractional limits and a charge with the prescribed ratio is subsequently prepared; for this reason it is necessary to assess the grindability of autoclave powders, for which batch I was used as an example (see Table 2). A rod mill (90 x 150 mm) rotating at 150 rpm was used for the experiments. The specific rod load ( $G$ ) and grinding time were varied while the powder weight ( $G_p = 50$  g) was kept constant. The bulk density and fineness of the powder increase and its specific surface decreases with an increase in grinding or in the rod mass (Fig.), but at  $\tau > 4-6$  min or  $G_r/G_p > 4$  these changes are not substantial, which is evidence of the rapid and easy grindability of the powder.

The pressability of powders processed by both schemes was then assessed. When the specific pressure increased to 8 tons/cm<sup>2</sup> the relative density of the briquettes reached 0.78-0.82 of theoretical; the pressability of the stabilized powder is somewhat inferior, the density of briquettes made from this powder in the 2-8 tons/cm<sup>2</sup> proving to be 6-7% less than in briquettes made from powder annealed in an atmosphere of hydrogen.

Briquettes made from powder stabilized with naphthenate soap (see Table 2, sample 1) were melted at 1130-1170°C in a neutral atmosphere. Blowing the powder into molten copper was also tested<sup>3</sup>. In both cases the chemical composition of the ingots produced met the requirements of GOST 859-66 for grade MO in terms of impurities content.

The present work has shown that organic additives have a substantial effect upon the properties of autoclave powder. Autoclave powder must obviously be treated in an atmosphere of hydrogen to improve its properties and purity if it is to be sold commercially or used for rolling.



Effect of duration (a,  $G_r/G_p = 8$  and rod load (b,  $\tau = 10$  min) upon average particle size (1), bulk density (2), and specific surface (3) of powder.

<sup>2</sup>The authors thank M. I. Nomberg and V. I. Romanovskii for their valuable assistance.

<sup>3</sup>The work was carried out by E. A. Yamshchikova and V. V. Podminogin.



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# A Proposed Method, Using Bacteria, for the Continuous Leaching of a Uranium Ore

H. H. McCREEDY, Research Scientist,  
V. F. HARRISON, Research Scientist,  
W. A. GOW, Head, Hydrometallurgy Section,  
Extraction Metallurgy Division, Mines Branch,  
Dept. of Energy, Mines and Resources,  
Ottawa, Ontario

## ABSTRACT

A semi-continuous bench-scale, countercurrent-decantation system, in which the ore was moved countercurrently to the flow of leaching solution, was used to conduct acid-leaching tests on uranium ore with the aid of bacteria. This method, which involved a low degree of agitation, was successfully applied to extract uranium in practical amounts and at practical rates. The acidity of the leaching solution was the main variable investigated in the bench-scale tests done at the Mines Branch, Ottawa. Over 90 per cent of the uranium was extracted in a six-stage system. The results suggest that this technique could be applied in the utilization of bacterial oxidation and leaching as a method for treating Elliot Lake uranium ore.

## INTRODUCTION

PREVIOUS WORK<sup>(1)</sup> has established the important part played by autotrophic bacteria of the *Ferrobacillus-Thiobacillus* group in the extraction of uranium from broken ore in worked-out stopes of the mines of Elliot Lake, Ontario. However, although uranium has been recovered from the Elliot Lake mines by this underground "bacterial leaching" technique, the amount of uranium produced by this method has been but a small proportion of the district's total uranium production. Each of two mines has reported a maximum daily  $U_3O_8$  production from underground leaching of 350 pounds<sup>(2,3)</sup>. This amount is equivalent to the production obtained by the conventional leaching of 175 tons of ore, as compared with the peak mining rate of 3,000 to 6,000 tons per day that is common in the Elliot Lake operations.

It is obvious from these comments that the production to date of uranium by underground "bacterial leaching" has been an interesting but very small sideline to the production derived from the conventional mining and milling techniques. Further consideration shows that underground bacterial leaching of primary ores is not likely to reach a greater prominence than it now enjoys. For example, in a recent investigation at the Mines Branch<sup>(4)</sup> of bacterial leaching of minus-14-mesh ore at a temperature of 13°C (the temperature of the stopes in the Elliot Lake mines), it was shown that the uranium ore would have to be exposed to the bacterial leaching solutions for from nine months to a year in order to extract 90 per cent of the contained uranium. This means that, under these conditions, about one million tons of minus-14-mesh ore would have to be undergoing bacterial leaching underground at any given time to ensure a uranium production rate equivalent to that of a 3,000-tpd conventional acid-leaching plant. In view of the unattractive prospects of successfully applying underground "bacterial leaching" methods to run-of-mine ore, the



McCREEDY

HARRISON

GOW

H. H. McCREEDY holds B.Sc. and M.Sc. degrees in mining and metallurgical engineering from the University of Alberta. He has been a member of the Mines Branch, Department of Energy, Mines and Resources, for the past fifteen years and currently holds the rank of research scientist in the Extraction Metallurgy Division.

VICTOR F. HARRISON received his B.Sc. degree in chemical engineering from Queen's University in 1943. After graduation, he served with Imperial Oil Ltd., the Royal Canadian Engineers and Queen's University before joining the Mines Branch of the Department of Energy, Mines and Resources in 1947. Since that time, he assisted in the development and application of hydrometallurgical processes for recovering uranium from Canadian ores. Recently, he has specialized in bacterial leaching of uranium ores, and has published several papers on this subject.

W. A. GOW was born in Toronto, and graduated with a B.A.Sc. in mining engineering from the University of Toronto in 1943. After serving three years with the Royal Canadian Engineers, he joined the Mines Branch of the Department of Energy, Mines and Resources in Ottawa in 1946, where he is at present head of the Hydrometallurgy Section of the Extraction Metallurgy Division. He had an active role from 1946 to 1959 in the work done at the Mines Branch in that period to develop the new technology required for the economic treatment of Canadian uranium deposits. From 1959 to the present, his work in the field of applied hydrometallurgical research has broadened to include many aspects of the treatment of Canadian mineral deposits in general.

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attention of Mines Branch hydrometallurgists was directed toward determining whether bacteria could be used in some other way to treat Elliot Lake run-of-mine ore.

In the current leaching practice in the Elliot Lake area, 60 to 80 lbs of  $H_2SO_4$  per ton of ore is used at a leaching temperature of  $70^\circ C$  to obtain 93 to 95 per cent extraction of the uranium in 72 to 96 hours of leaching. The cost of acid and heat for this process ranges from about \$1.40 to \$1.90 per ton of ore. It is also known<sup>(5)</sup> that uranium extractions of 91 or 92 per cent can be obtained by leaching the Elliot Lake ores at a pH of 1.5 and at a temperature of 25 to  $30^\circ C$  in 96 hours, provided that the oxidizing potential of the leach solution is maintained at 500 mv or higher<sup>(6)</sup>. The cost of acid and heat for this second set of leaching conditions would be about \$0.50 per ton if the cost of oxidizing the leaching solution could be kept low. This would represent a saving, over the cost of current practice, of from \$0.90 to \$1.40. As one of the functions of the *Ferrobacillus-Ferrooxidans* bacteria is to oxidize ferrous iron into ferric iron, the advantages of the low-acid leach described above might be realized if bacteria could be used to provide the required oxidizing condition in the leach solution.

To test this possibility, a series of tests was carried out at the Mines Branch in which the ore was leached in beakers at various pulp densities, with bacteria added to oxidize the leaching solution<sup>(4)</sup>. In that work, the slurry was stirred vigorously during the test. The results were disappointing, because the bacterial action died off at pulp densities of 35 per cent or over. As it is not economical, because of the resulting high capital and operating costs, to leach the Elliot Lake ores in agitators at less than 35 per cent solids, this approach to the practical use of bacteria was abandoned.

When it was apparent that conventional agitator leaching was unsuitable for the utilization of the oxidizing characteristics of the bacteria in a low-acid leaching process, attention was turned to another leaching technique, proposed by Mines Branch staff in 1958<sup>(6)</sup>. In that earlier proposal, it was suggested, on the basis of laboratory results, that a conventional countercurrent-decantation system, using several thick-

eners could be used, not only to recover the uranium-bearing solution subsequent to leaching, but also to carry out the leaching operation simultaneously with the liquid-solid separation. In that 1958 work of Brown *et al.*, the thickener underflows and overflows flowed countercurrently to each other, and between the thickener stages a short agitation step was provided during which the acidity of the slurry could be adjusted as required.

The conditions in a thickener, including an almost clear supernatant solution in the upper portion and a gently agitated slurry in the bottom portion, appeared to be suitable for bacterial action to take place. Consequently, the feasibility of applying the proposed method of Brown *et al.* to leaching the Elliot Lake ore with the aid of bacteria was investigated by Mines Branch staff. This paper, which should be thought of as a progress report, describes the results of the first stage of this investigation, in which small-scale laboratory studies simulating a semi-continuous countercurrent-decantation system were conducted. The results of this work are given and discussed.

## PROCEDURE

The ore used as feed material was representative of the mill feed to one of the operating mines in the Elliot Lake area. The chemical analysis of this material is given in Table I and shows that the ore contained 0.089 per cent  $U_3O_8$ . Uraninite and brannerite were the major uranium-bearing minerals present, but minor amounts of monazite, coffinite and thucolite were also present. The sulphur content was almost entirely pyrite, although a minor amount of pyrrhotite was also present<sup>(7,8)</sup>. The leach feed was ground to about 65 per cent minus 200 mesh (Table II).

A six-stage, countercurrent-decantation (CCD) system was simulated in this study, using 2-litre graduated cylinders to represent the thickeners of a full-scale, continuous CCD, wash-thickening operation. In addition to the graduated cylinders, the test apparatus included an aerated vessel in which bacterially active, highly oxidized Silverman 9K solution (Table III) was prepared for use in the leaching circuit. All of the apparatus was housed in an enclosed cabinet in which the temperature was controlled automatically to  $32^\circ C$ .

In the tests where the slurry was aerated, the air was introduced through a fritted-glass, air dispersion tube to the upper one-third of the solution in the cylinders. To reduce evaporation and heat losses due to aeration, the air was heated to the leaching temperature and saturated with water vapour before it was introduced to the leaching vessels.

Table I—Chemical Analysis of Leach Feed

Analysis	Per Cent
$U_3O_8$ .....	0.089
$CO_2$ .....	0.14
Fe.....	3.10
S.....	2.27
P.....	0.02

Table II—Screen Analysis of Leach Feed

Mesh Size	Weight, per cent
+ 48.....	0.1
- 48 + 65.....	0.4
- 65 + 100.....	8.3
- 100 + 150.....	12.8
- 150 + 200.....	13.5
- 200.....	64.9
	100.0

Table III—Composition of Unoxidized Silverman 9K Medium\*

Constituent	Concentration, g/l
$FeSO_4 \cdot 7H_2O$ .....	44.2
$(NH_4)_2SO_4$ .....	3.0
$K_2HPO_4$ .....	0.5
KCl.....	0.1
$MgSO_4 \cdot 7H_2O$ .....	0.5
$Ca(NO_3)_2 \cdot 4H_2O$ .....	0.02
$H_2SO_4$ .....	to desired pH

\*Reference 9.

Table IV — Solution Analyses in the Simulated CCD Circuit

	Fresh Leach Solution	→				←		New Ore
		Stage 6	Stage 5	Stage 4	Stage 3	Stage 2	Stage 1	
pH.....	2.00	1.97	2.09	2.22	2.28	2.30	2.30	
EMF, mv.....	570	552	553	559	554	556	555	
Fe <sup>+2</sup> , g/l.....	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Fe tot., g/l.....	8.4	7.9	7.5	7.1	6.6	6.0	6.1	
pH.....	1.75	1.79	1.81	1.86	1.95	2.07	2.16	
EMF, mv.....	544	539	539	546	554	552	552	
Fe <sup>+2</sup> , g/l.....	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Fe tot., g/l.....	8.8	8.3	7.8	7.7	7.2	6.9	7.0	
pH.....	1.60	1.71	1.64	1.65	1.72	1.84	2.00	
EMF, mv.....	493	527	516	503	534	551	550	
Fe <sup>+2</sup> , g/l.....	1.0	0.1	0.2	0.3	0.1	0.0	0.0	
Fe tot., g/l.....	8.9	8.2	8.3	8.1	8.1	7.9	7.4	
pH.....	1.50	1.54	1.54	1.63	1.78	1.92	2.05	
EMF, mv.....	456	483	488	500	534	550	552	
Fe <sup>+2</sup> , g/l.....	2.0	0.68	0.39	0.24	0.0	0.0	0.0	
Fe tot., g/l.....	8.8	8.7	8.3	8.3	8.1	7.9	7.8	
pH.....	1.40	1.41	1.49	1.63	1.77	1.90	1.97	
EMF, mv.....	419	430	443	473	542	558	556	
Fe <sup>+2</sup> , g/l.....	3.8	2.9	1.8	0.61	0.0	0.0	0.0	
Fe tot., g/l.....	8.7	8.8	8.5	8.4	7.8	7.8	7.8	

When the system was operating, fresh ore was introduced to stage 6 and leached residue left the system at stage 6, while fresh leaching solution entered at stage 6 and pregnant uranium-bearing solution left at stage 1. In each stage, the pulp was agitated briefly at a density of 30 per cent solids and then allowed to settle. The fresh make-up solution added to stage 6 was completely oxidized Silverman 9K solution acidified to the pH value being investigated at the time, and containing active bacteria obtained originally from mine water from an Elliot Lake mine. The pulp in each stage was then allowed to settle and, at 24-hour intervals, the supernatant solution from each stage was removed and advanced one stage while the settled slurry, at about 60 per cent solids, was moved one stage in the opposite direction. The solution and pulp transferred to each stage were then mixed briefly as before and allowed to settle again. This procedure resulted in the ore being held for twenty-four hours in each of six stages — a total of one hundred and forty-four hours. In this procedure, the only acid added to the entire CCD system was that added to the fresh, bacteria-bearing solution added to stage 6. Each day, the residue, after having been washed twice with dilute acid and once with water, was analyzed chemically for uranium, as was the pregnant solution from stage 1.

Under suitable conditions, bacteria present in an iron sulphate solution will maintain nearly all of the iron in the ferric state, which in turn results in a solution having a high oxidizing potential. Consequently, the bacterial activity in this CCD system was monitored by measuring daily the ferrous and ferric iron concentrations and the pH and redox values of all the solutions produced — the latter two determinations were made with reference to a calomel electrode.

The data reported in this paper are the results of operating the system continuously for about ten weeks. During this time, pH values in the range of 1.4 to 2.0 were investigated. The effect of increasing

the leaching time to eight days by the addition of two stages was also studied.

## RESULTS

At the start of the test program, the leaching vessels were not aerated. The effect of this was to reduce bacterial activity in the leaching vessels to such a degree that the reactions between the ore and the solution reduced the solution faster than the bacteria could oxidize it. The consequence of this was that after twenty-four hours about 40 per cent of the iron in solution had been reduced to the ferrous state. As it was known that the successful leaching of the Elliot Lake ores at pH values of 1.5 to 2.0 required a very low ferrous-iron concentration, air was bubbled into the leaching vessels to increase the bacterial activity to the point where it would prevent a build-up in the ferrous-iron concentration of the solutions. For the duration of the test program, 700 millilitres of air per minute was added to each leaching vessel. This amount was found to be sufficient, but no work was done to determine the minimum amount of air required.

Earlier percolator-type experiments to investigate the leaching of Elliot Lake ore with the help of bacteria<sup>(6)</sup>, as well as earlier controlled-pH leach tests<sup>(5)</sup>, showed that to obtain satisfactory uranium leaching rates the leaching solution should have a pH value of about 1.5. Other investigators had shown that maximum bacterial activity occurred in the pH range of 2.0 to 2.5. Because of this background information, it was decided to study, in this test program, how changing the pH of the leaching solution from 2.0 to 1.4 affected the bacterial activity and the uranium leaching rate. The pH was adjusted in the aerated vessel where the highly oxidized, bacterially-active Silverman 9K solution was prepared prior to its addition to the leaching vessels. This was mentioned in describing the general test procedure. No acid was added to the leaching vessels. The results of this study are given in Figure 1 and Table IV.

Figure 1 shows that as the pH of the fresh solution was decreased from 2.0 to 1.6, the uranium extraction obtained in six days of leaching increased from 84.6 to 91.1 per cent. A further decrease in the pH resulted in a decrease in uranium extraction from the 91.1 per cent obtained at pH 1.6 to 88.8 per cent at pH 1.4.

The reason for this behaviour is apparent from the data in Table IV. At pH values down to 1.6, the degree of acidity does not reduce bacterial activity to a point where ferrous iron is present in any of the leaching vessels. Consequently, the oxidizing conditions at pH 1.6 were the same throughout all six leach stages as they were at pH 2.0, and the increased acidity at pH 1.6 resulted in a faster extraction rate. At pH values of less than 1.6, the acidity level reduces bacterial activity to such an extent that the ferrous-iron content of the leaching solution is high and its oxidizing potential is low during the first three stages of leaching. Apparently, this low oxidizing potential in stages 6, 5 and 4 outweighs any benefit due to the greater acidity at the lower pH values, so that the net result is a decrease in the uranium extraction rate.

When it was found that the highest uranium extraction was obtained when the leaching solution had a pH of 1.6 at the time it was introduced to the leaching circuit, the effect of increased leaching time was examined. This was done by adding two extra stages (stages 7 and 8) of 24 hours each to the CCD system. No further extraction of uranium was obtained in the two additional stages, indicating that the six-stage system was sufficient to achieve ultimate extraction at the grind used.

A graph of the uranium extraction versus the logarithm of the retention time for an eight-day leaching operation, when the fresh leaching solution had a pH of 1.6, is given in Figure 2. This graph shows that the extraction rate under these conditions is exponential, as was the extraction rate observed in earlier work where 14- to 20-mesh ore was leached in the presence of bacteria in percolators<sup>(8)</sup>. Also, the rate constants in the present work and in that earlier work

are similar, both being in the range of 32 to 53 when calculated from the units shown in Figure 2.

The amount of sulphuric acid required to adjust the leaching solution to the various pH values investigated is given in Figure 3. This figure shows that it required 23.5 pounds of H<sub>2</sub>SO<sub>4</sub> per ton of ore to adjust the pH of the fresh leaching solution to 1.6. Chemical analyses for sulphur, done on the leach feed and the leach residue, showed that only about 2 pounds of sulphur per ton of ore was extracted from the ore and that this extraction was independent of the pH of the leach. Even if all of this sulphur was converted to sulphuric acid, it would be well short of the 23.5 pounds needed for the necessary pH adjustment. These results indicate that bacterial attack on the sulphide minerals is too slow to produce a significant amount of acid in the six days during which the ore is in contact with the bacteria-bearing solutions.

## DISCUSSION

This bench-scale work has shown that semi-continuous CCD acid leaching at pH 1.6, with bacteria present to assist in the required oxidation reaction, is chemically and technically feasible. Further confirmation of the technical feasibility is the CCD system of wash thickeners that has been used successfully at the Algom Quirke mine in the Elliot Lake area for efficient solution recovery. However, the method used in our bench-scale work requires study on a continuous pilot-plant scale to clarify the economics of the proposed process and to determine whether it is economically viable.

For example, in this bench-scale work, Silverman 9K solution was used as the fresh solution make-up to the system. It would, of course, be economically unrealistic to use this solution in a full-scale plant, because of the high reagent cost involved in its preparation. However, the solution produced in the acid leaching of Elliot Lake ores is similar in composition to the 9K medium and, if it is recycled to the CCD leaching step after it has been treated for uranium recovery, it should be capable of supporting adequate

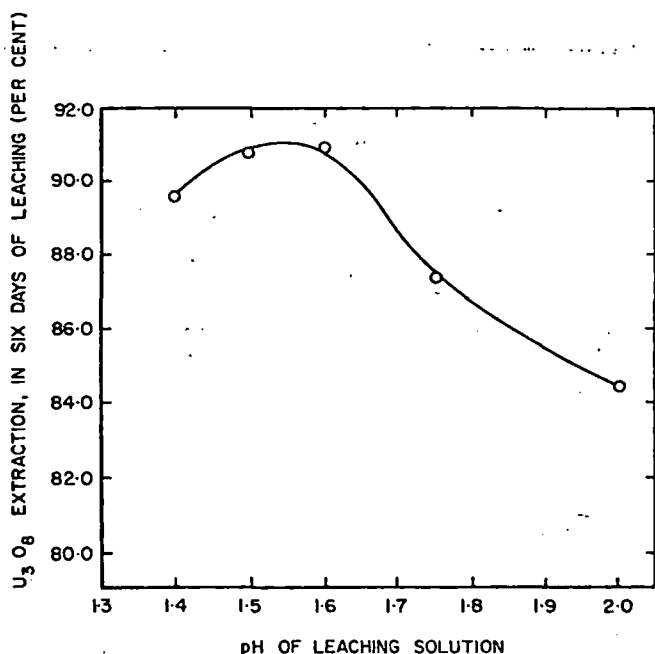


Figure 1.—Effect of the pH of the Leaching Solution on Uranium Extraction.

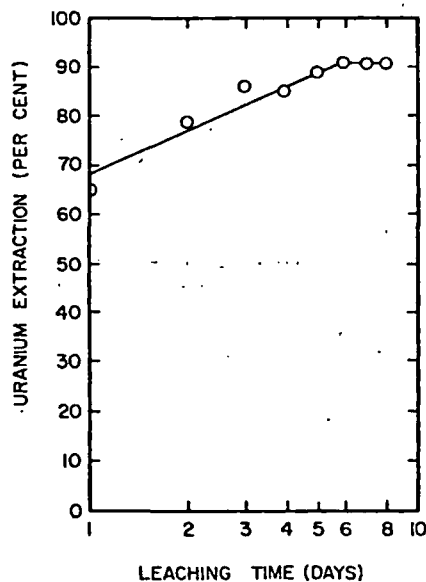


Figure 2.—Uranium Extraction at a pH of 1.6.

**Table V — Comparison of Unoxidized Silverman 9K Solution and Elliot Lake Recycled Leach Liquor, g/l**

Element	9K	Leach Liquor*
Fe (total)	8.9	16.3
Fe <sup>+2</sup>	8.9	11.8
(NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub>	3.0	—
KCl	0.1	—
MgSO <sub>4</sub> · 7H <sub>2</sub> O	0.5	—
Co(NO <sub>3</sub> ) <sub>2</sub> · 4H <sub>2</sub> O	0.02	—
P <sub>2</sub> O <sub>5</sub>	0.2	0.4

\*From Special Report SR 359/55 of the Radioactivity Division, Mines Branch (1955).

bacterial activity (Table V). It has, in fact, been shown that the barren effluent from an Elliot Lake ion-exchange operation is a suitable medium for bacterial growth<sup>(1)</sup>. A further indication that recycling the leaching solution in this proposed process is feasible is that it has been shown by Harrison *et al.*<sup>(2)</sup> that, when Elliot Lake ore ground finer than 8 mesh was leached with the help of bacteria, the addition of nitrogen, phosphorus and potassium nutrients was not required to maintain a good uranium extraction rate.

It is also necessary to determine how many thickening stages will be required in a continuously operating system. By-passing of the solids through the thickeners could result in an uneconomical number of stages in the CCD system. On the other hand, the gentle agitation of the settled solids in an operating thickener may increase the uranium extraction rate over that obtained in the almost static conditions of the test work just completed. The effect of the uranium recovery step, whether it be an ion-exchange or solvent-extraction operation, on the bacterial activity in the recycled solution will also have to be determined.

Although we still lack a great deal of the information necessary to fully assess the value of a CCD

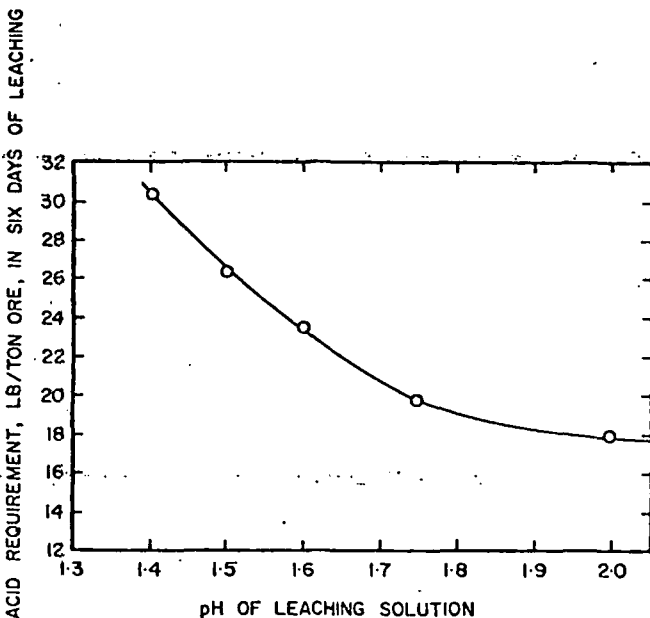


Figure 3.—Effect of the pH of the Leaching Solution on Acid Consumption.

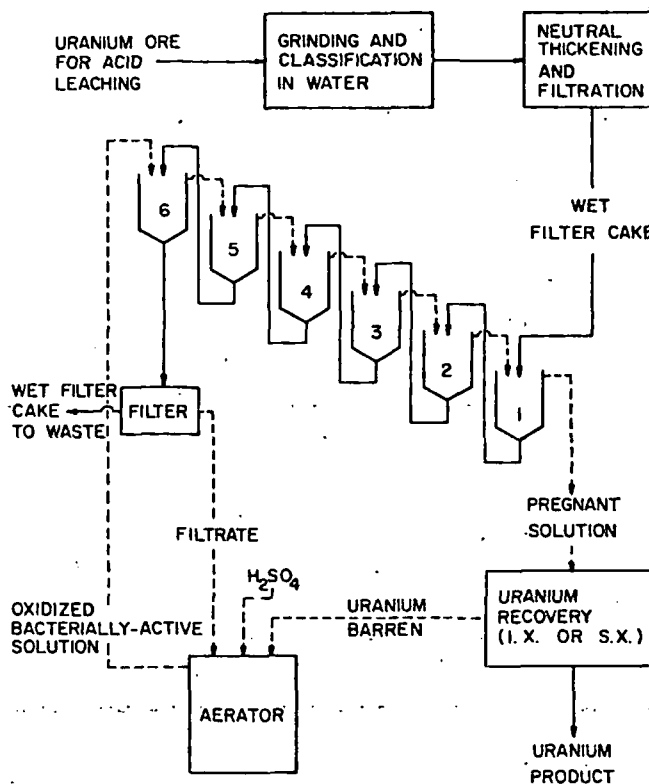


Figure 4.—Proposed Flow Sheet for Leaching Elliot Lake Ore with the Help of Bacteria.

uranium-leaching circuit in which bacteria are used to provide the oxidation conditions required, we think the results of the bench-scale laboratory work indicate that the process could be an attractive alternative to the conventional acid-leaching process. A process similar to that shown in Figure 4 would not only require a lower reagent consumption than the process now in use, but the elimination of agitators would reduce the capital cost of the plant. The capital saving results from the fact that, at present, agitators are used for leaching along with a CCD circuit, or several stages of filtering to recover the uranium-bearing solution, whereas in the proposed process, the CCD circuit is used both for leaching and for solution recovery.

The proposed flow sheet, which we intend to test on a pilot-plant scale, involves grinding of the ore in water to 65 per cent minus 200 mesh, followed by thickening and filtering. The filter cake from the grinding circuit would then be fed to the first of six thickeners. The flow of the ore would be countercurrent to the flow of bacterially active, highly oxidized, ferric-iron leaching solution. The uranium-bearing solution, which would be the overflow from stage 1 of the CCD circuit, would be treated by ion exchange or solvent extraction to recover the uranium. The barren from the uranium recovery step would then be aerated for 24 to 48 hours to generate a well-oxidized, bacterially active solution, after which it would be returned to stage 6. The leached slurry from stage 6 would be filtered, with the filtrate going to the aerator and the filter cake going to waste. The solution in this filter cake would be the main solution bleed from the circuit, balancing the water entering with the leach feed. If additional bleed should be necessary, some of the barren from the uranium recovery step could be sent to waste instead of to the aerator.

## CONCLUSIONS

The following conclusions can be drawn from the results of the bench-scale test work reported in this paper:

- (1) A CCD system of simultaneous leaching and settling, in which bacterial action is used to maintain the required oxidizing conditions in the leach solutions, has a very good chance of proving to be an economical method for the extraction of uranium from its ores.
- (2) A CCD system such as described above would result in a uranium extraction, from ores similar to those of the Elliot Lake area, of over 90 per cent, with an acid consumption of about 25 lbs H<sub>2</sub>SO<sub>4</sub> per ton of ore at 30°C. This compares favourably with the conventional acid-leaching process, which involves an acid consumption of 60 to 80 lbs H<sub>2</sub>SO<sub>4</sub> per ton at 70°C.
- (3) The cost of constructing and operating a continuous CCD pilot plant to further evaluate the proposed process is warranted.

## ACKNOWLEDGMENTS

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## REFERENCES

- (1) V. F. Harrison, W. A. Gow and K. C. Iverson, "Leaching of Uranium from Elliot Lake Ore in the Presence of Bacteria," *Can. Min. J.*, 87, No. 5, 64-67 (1966).
- (2) R. A. MacGregor, "Recovery of U<sub>3</sub>O<sub>8</sub> by Underground Leaching," *CIM Bulletin*, Vol. 59, No. 649, 583-587; *CIM Trans.*, Vol. LXIX, 162-166; (1966).
- (3) J. R. Fisher, "Bacterial Leaching of Elliot Lake Uranium Ore," *CIM Bulletin*, Vol. 59, No. 649, 588-592; *CIM Trans.*, Vol. LXIX, 167-171; (1966).
- (4) V. F. Harrison — unpublished data.
- (5) B. H. Lucas and W. A. Gow, "Controlled pH Leaching of Elliot Lake Ores," Mines Branch Investigation Report IR 60-96, Dept. of Energy, Mines and Resources, Ottawa, p. 1 (1960).
- (6) E. A. Brown, et al., "Some Variations of Uranium Ore Treatment Procedures," *Peaceful Uses of Atomic Energy* (United Nations Publications, Geneva), 3, 195 (1958).
- (7) W. R. Honeywell and S. Kaiman, "Flotation of Uranium from Elliot Lake Ores," *CIM Bulletin*, Vol. 59, No. 647, 347-355; *CIM Trans.*, Vol. LXIX, 99-107; (1966).
- (8) V. F. Harrison, W. A. Gow and M. R. Hughson, "Factors Influencing the Application of Bacterial Leaching to a Canadian Uranium Ore," *J. Met.*, 18, 1189-1194 (1966).
- (9) M. P. Silverman and D. G. Lundgren, "Studies on the Chemoautotrophic Iron Bacterium *Ferrobacillus Ferrooxidans*," *J. Bacteriol.*, 77, 642-647 (1959).

## AIME Announces 1969 Award Winners

THE AMERICAN INSTITUTE OF MINING, METALLURGICAL, AND PETROLEUM ENGINEERS have announced that the following are the recipients of their 1969 awards:

DR. CHARLES J. FOGARTY, president and director of Texas Gulf Sulphur Company . . . . *Hal Williams Hardinge Award*

B. J. SNYDER, general foreman, blast furnaces, Armco Steel Corporation . . . . *J. E. Johnson, Jr. Award*

DR. JAMES H. SWISHER, research engineer, Bell Telephone Laboratories . . . . *Rossiter W. Raymond Memorial Award*

J. HALL CARPENTER, president and chairman of the board of Carpc Research and Engineering, Inc. . . . . *Robert H. Richards Award*

S. K. W. MORGAN, managing director of Imperial Smelting Processes . . . . *James Douglas Gold Medal*

C. D. MICHAELSON, president of the Metal Mining Division of Kennecott Copper Corporation . . . . *Daniel C. Jackling Award*

A. M. GAUDIN, Professor of Mineral Engineering at the Massachusetts Institute of Technology . . . . *Mineral Industry Education Award*

CLARENCE J. COBERLY, honorary chairman of the board of Kobe, Inc. . . . . *Anthony F. Lucas Gold Medal*

DR. JAMES B. AUSTIN, retired administrative vice-president for research and technology at U.S. Steel Corporation . . . . *Benjamin F. Fairless Award*

DR. JAMES H. PIERCE, chairman of the board of Pierce Management Corporation . . . . *Erskine Ramsay Gold Medal*

WILLIAM PAGE MORRIS, president and director of Duval Corporation . . . . *Charles F. Rand Memorial Award*

Hope may not be dead for a famous defunct copper district. A metallurgist sees way through new technology of extracting large remaining mineral reserves.

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# Ammonia Revival for the Keweenaw?

R. S. Shoemaker

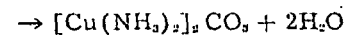
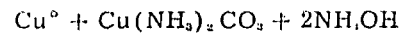
Mines are closed for two reasons; exhaustion of ore or insufficient price for the mineral. On the other hand, the reopening of an old mine can be the result of any one of three events; the discovery of more ore, a sudden rise in the price of the product or the use of a new low-cost method of mining or milling the ore. Examples of the latter would be the open-pitting of an orebody previously mined underground as recently happened at Tyrone, N.M., or the adoption of acid leaching of an old pit as has been done at Miami, Ariz.

The unique native copper mines of the Keweenaw Peninsula of upper Michigan have not been so fortunate. These deposits, which were first worked by prehistoric Indians, have been mined continuously from the early 1840's to 1968 when the last mines of Calumet and Hecla were closed and the Ahmeek mill, still using eight Nordberg steam stamps, was shut down. There are two types of deposits from which native copper is obtained; amygdaloid, which is a lava flow, and conglomerate, a sedimentary deposit of sand, rock fragments and pebbles cemented together. "In the amygdaloid, the copper is disseminated, or distributed much like the raisins in a cake; but in the rich Calumet conglomerate, the copper is more like a malleable rope or thread cementing the pebbles together." (C. Harry Benedict. *Red Metal: The Calumet and Hecla Story*, (Ann Arbor, Michigan, 1952), p. 42.) These deposits produced almost all of the copper used by the North in the Civil War and were substantial contributors to the copper needs of the United States in both World Wars. Some were mined to depths of as much as 9500 ft along the dip of the vein structure and 7500 ft vertically. Few of the mines ever really became worked out, but became uneconomical because of depth, pinching of veins or grades which dropped to about 1%.

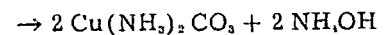
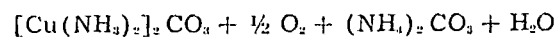
Although copper prices have been in the neighborhood of 60¢ per lb in the last few years and forecasters are projecting prices in excess of that in the future, the above reasons create doubt that the area will ever be rejuvenated as a major mining district through conventional methods despite the fact that over 10 billion lb of copper have been recovered and probably as much more remains in the so-called mined-out areas. This copper is present in shaft

pillars, stope pillars, old stopes filled with low grade and low grade zones on both the hanging and footwalls. Additional copper is certainly present in low grade unmined areas.

Since high copper prices cannot possibly overcome the high costs of conventional mining methods and the complex milling methods needed to process the ore, then perhaps what is needed is an entirely new approach to the problem. This approach may already exist in the form of a patent granted to C. H. Benedict on March 16, 1915, which described a leaching process using cupric ammonium carbonate as the lixiviant. Benedict was chief metallurgist of Calumet and Hecla and his invention of this process not only made it possible to treat lower grade native copper ores than had been treated before, but also made it feasible to reprocess millions of tons of tailings which had accumulated for over fifty years. The process produced copper at such a low cost that it was the determining factor in maintaining the economic existence of the company during the great depression of the Thirties. Briefly, a sand fraction which was classified from the recovered tailing (the slimes were subjected to flotation) was subjected to downward percolation leaching at atmospheric pressure with the solution reaction being as follows:



The cuprous solution formed was then reoxidized by merely contacting it with air countercurrently in a tower containing grids. Oxidation proceeded rapidly in accordance with the following reaction to produce more leaching agent:



Carbon dioxide, the other reagent, was introduced along with the air in the oxidation towers.

Copper was recovered as a mixed oxide by boiling the pregnant solution and the ammonia was recovered for recycling. Acid was not used for leaching both because an oxidizing reaction was necessary and the ore contained a large amount of calcite. The ammonia leach had an additional advantage of pro-





In the early days masses of copper like this one from the 79th level of the Quincy weighing 15 tons were common. The largest ever found weighed over 500 tons. Photo courtesy of Angus Murdock.

ducing a high purity pregnant solution since none of the gangue minerals would react with the leach solution, and metals which would become impurities in the recovered copper were either not present in the ore or did not dissolve. The minor amount of native silver in the ore was also left reacted.

Since the Benedict process has been proved in practice and enormous reserves of copper are known to exist in the old mines, it would seem that the two could be brought together in the form of *in situ* leaching, along with other developments in hydrometallurgy, for some interesting results. The ore bodies of the Keweenaw Peninsula are located in a few of a great number of lava flows and sedimentary beds which dip rather steeply near the surface but considerably less at depth. The veins were relatively narrow and generally less than 20 ft wide, with some of them only 4-5 ft wide. Additionally, often only the center and richest portions of the veins were mined, partly because of the higher grade and partly because the ore was stronger than the wall rock. To mine the ore full width would have permitted the footwalls to swell and the hanging wall to cave. This method of mining left substantial values in addition to those in the shaft and floor pillars and in the old stopes which were often filled with material too low grade to be called ore at the time. With subsequent caving of backs and crushing of pillars as retreating took place, it is reasonable to assume that quite a porous structure having an enormous surface area suitable for percolation leaching would remain between relatively impervious rock confines which could be likened to the walls of a leaching vat or autoclave.

### First Pumping, Then Leaching

Most of the native copper mines were relatively dry. Much of the water that did come into the mines was surface water which entered along the contact between the alluvial cover and the bedrock. For instance, by collecting this surface water underground and draining it into Portage Lake through an adit which intersected the shaft, Quincy Mining Co. was able to keep their 9600 ft deep mine dry by bailing out the small inflow with a bucket attached to the hoist rope during the night shift. Of course, after many years of being closed, the mines will have filled with water, but modern grouting techniques could be used to seal off the surface water and after the big job of initial pumping, leaching could be carried out and dilution controlled by discarding at the surface a portion of the leach solution which had been stripped of its copper and ammonia content in the recovery plant.

Preparation for leaching after pumping might involve driving a drift along the vein through the old workings, perhaps 100 ft below the surface in which the leach solution distribution piping could be laid. At the same time, old shafts and stopes which might have broken through to the surface could either be sealed off or equipped with scrubbers for ammonia recovery. Most probably the lower reaches of the mines would be inaccessible from the surface, but existing maps would permit drilling to intersect the lowest levels. Two drill holes might be sufficient, one through which to pump the pregnant solution to the surface and the other for pumping oxygen and carbon dioxide down into the mine. The  $O_2$  and  $CO_2$

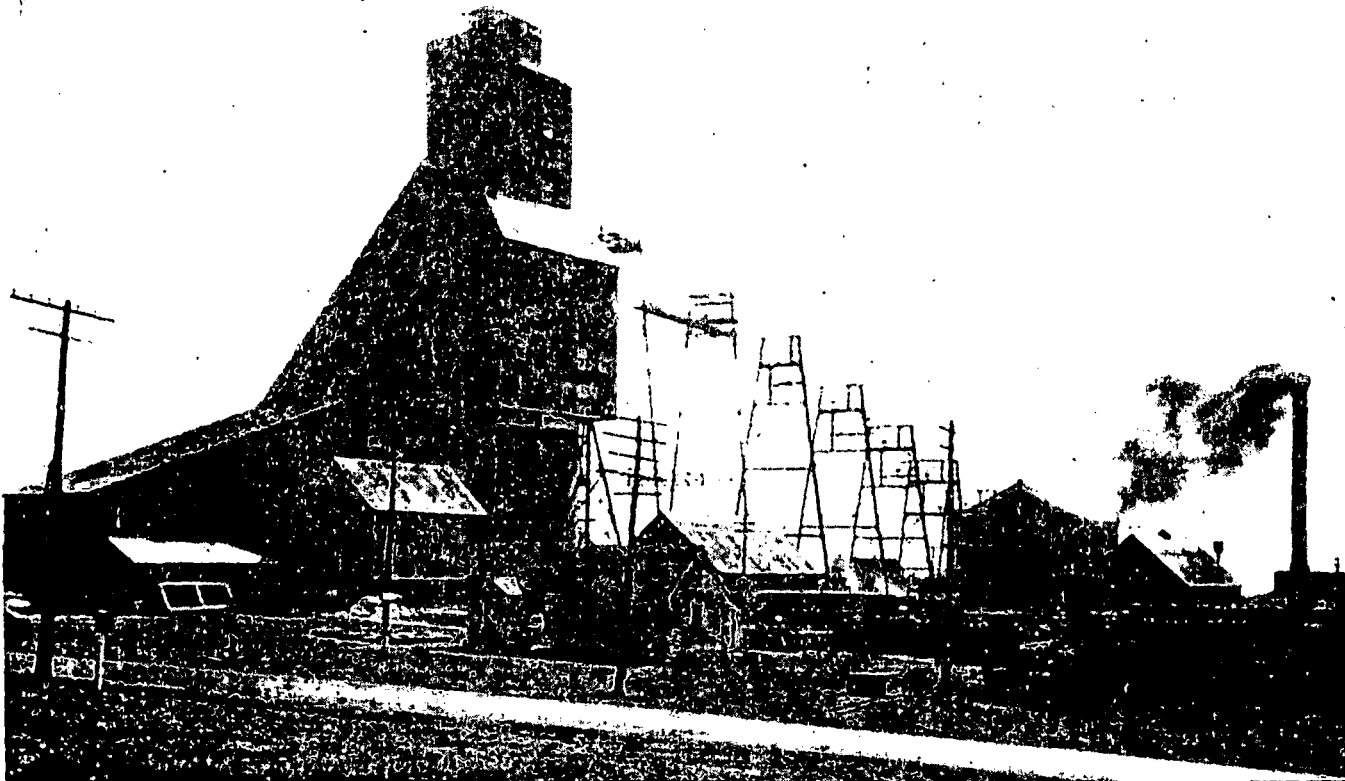


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R. S. Shoemaker  
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The Quincy No. 2 shaft house, hoist and boiler. All of these structures are still standing. The hoist, which served the 9600-ft inclined shaft is now a museum.

would completely fill the mine, so that oxidation of cuprous ion to cupric would take place immediately as the cuprous was formed, and so the correct ammonium carbonate level could be maintained. The result would be an enormous leaching chamber containing uncounted tons of ore in the hanging and footwalls, pillars, etc. Benedict, in his book *Red Metal* quotes the USBM as stating that the Kearsarge lode had the greatest area of unsupported openings of any mine in the world. As the copper was removed, thus weakening the pillars and backs, more crushing could take place which would continually expose more copper. Since percolation leaching would be used, the amount of leach solution would be kept to a minimum.

Instead of recovering the copper as oxide by boiling the solution, as was done by Calumet and Hecla, a portion of that solution pumped from the mine would be treated by solvent extraction and electro-winning. The remainder would be trickled into the top of the mine as leaching solution. Pollution of groundwater should not be a problem since some water would continually flow into the mine. Pollution of the atmosphere should not occur either since the leaching area would not be under pressure. Ammonia and oxygen would be continually complexing with the copper and additions of these reagents would merely replace those used in the complexing reaction. Any openings needed for later access into the mine would be sealed and scrubbers could take care of ammonia escaping under unusual conditions. Any

buildup of leach solution through in-leakage of water would be taken care of by bleeding off a portion of recovery plant raffinate and processing it through an ammonia recovery still before discarding it.

Surface facilities would be composed of oxygen and carbon dioxide plants, a pump house, a solvent extraction plant and an electro-winning plant. Carbon dioxide would be generated by burning oil or gas, and the heat produced would be used to heat the surface facilities in the winter time. For each pound of copper produced, approximately 0.3 lb of oxygen and 6 cu ft of natural gas would be required. Since the ammonia would be recovered, only the initial charge and small make-up quantities would be required.

Since recoveries would be difficult to forecast, it is suggested that one of the smaller, shallower mines which is isolated from its neighbors be converted into a research laboratory.

In considering the foregoing, it is possible for the entire copper-bearing vein structure of the Keweenaw Peninsula, which extends for some hundred odd miles into Lake Superior, to become one huge *in situ* leaching field. Of course, some of the unmined portions of the conglomerate veins might not be porous enough for leaching solutions to penetrate easily, however, the amygdaloid veins and the ash beds where some of the richest ores were found might be. The use of hydrofracturing techniques, solvent extraction and electro-winning might be what is required to make the Upper Michigan native copper mines rise again from their ashes. Appropriately enough, one of those mines was named the Phoenix. **ME**

R. S. Shoemaker, an SME Member, is Consulting Metallurgist, Bechtel Corporation, San Francisco.

# minerals industry news

## Anaconda Will Develop New Nevada Open Pit Cu Mine

Eastern Nevada will have a new open-pit copper mine, according to John B. M. Place, president and chief executive officer of the Anaconda Co.

The new operation, to be designated the Victoria mine, will be sited in the Antelope Mountain range west of the Utah border and some 80 mi northeast of Ely, Nev.

Place explained that the project was made feasible by Anaconda's development of the Arbiter hydrometallurgical process (see *ME*, May '73 p. 14). He added that the Victoria property which has been held by the company since the mid '40's, contains ores high in bismuth content which could not be treated satisfactorily in conventional smelters and refineries. The Arbiter process would eliminate this problem. An extension will be added to the company's Anaconda, Mont., hydrometallurgy facility now under construction to accommodate treatment of Victoria concentrates via the Arbiter process.

Plans include construction of a concentrator by 1975 with a capacity to treat 1000 tpd ore.

Initial production from the property has been set at 9000 tpy finished copper. This figure is expected to increase yearly to a rate of 6000 tpy in 1984.

Anaconda estimated that the ore deposit contains about 3.5 million tons of copper sulfide ore with an average grade of 2.45%. By-products will include bismuth, silver and gold.

## Newmont, O'okiep Copper Disclose Zn Ore in South Africa

Newmont Mining Corp. and O'okiep Copper Corp. Ltd. have revealed that a drilling program conducted over the past year by an equally-owned joint venture of the two companies has uncovered what has been termed a large mineral deposit containing zinc sulfide at Gamsberg, about 80 mi east of O'okiep in Namaqualand, South Africa.

Gossan associated with the mineralization has been traced over a strike length of about 15,000 ft. Drilling done thus far over a portion of this length has yielded significant intersections of 30-100 ft in width of sulfides assaying out at between six and ten percent zinc and less than one half of one percent lead. If the

deposit is mined, underground methods would be employed.

The zinc mineral, which is believed to be marmatitic, is associated with iron sulfides consisting mostly of pyrite and pyrrolite, which may present metallurgical problems. Metallurgical testing is underway and drilling is continuing in order to delineate the orebody and determine average grade of ore.

According to Newmont spokesmen, it is probable that the deposit will have to support a zinc refinery as well as a mine and mill.

## Ranchers Development Sets Off Blast; Will Leach at Big Mike

Approximately 550,000 tons of copper ore and rock were blasted into an open pit mine near Winnemucca, Nev., July 10, with 400,000 lb of explosive to initiate what has been called "a unique copper leaching operation."

The blast was set off at Ranchers Exploration and Development Corp.'s Big Mike Copper Mine located about 30 mi south of Winnemucca. Personnel from DuPont Corp. were in charge of emplacing and detonating the explosive.

The explosive, consisting of ammonium nitrate pellets and water-proof slurry gel, was used to shatter about 325,000 tons of copper ore located in the walls of the open pit and about 75,000 tons of ore in the bottom of the 300 ft deep pit.

In addition to the ore, which contains approximately two percent copper, about 150,000 tons of waste rock were shattered by the blast.

Most of the explosive used in the blast was emplaced in some 175 drill holes located at various points around the perimeter of the pit. The holes, some of them 300 ft deep, were as far as 50 ft from the edge of the pit.

The remaining explosive was emplaced in approximately 75 holes in the bottom of the pit. Maximum depth of these holes was 100 ft.

The shattered ore and waste will be leveled and terraced in preparation for leaching, expected to begin later this month. Leaching will be carried out by sprinkling a solution of sulfuric acid and water on the core at the rate of about 200 gpm from ordinary lawn sprinklers.

This solution will contain approximately one percent acid. The copper-bearing liquid will then be pumped from the bottom of the ore heaps.

The pump will be located in a pipe-encased hole sunk about 150 ft below the top of the ore heap. The leaching solutions will drain into the hole through a perforated casing at the rate of about 200 gpm, then be pumped to a precipitation plant on the surface.

The operation—the first of its kind to produce copper by blasting ore into an open pit for heap leaching is expected to produce about 5000 lb of copper per day for three years.

The mine is currently producing about 5000 lb of copper cement daily from ore heaps which were created when some 95,000 tons of high grade ore were removed from the open pit in 1970 and shipped abroad. About 275,000 tons of lower grade ore were placed in heaps near the pit and have been under leach the past several years.

It was in March of 1972 that Ranchers detonated four million lb of explosive to crush a four million ton copper orebody at the Old Reliable mine near Mammoth, Ariz. (see *ME*, May '72 p. 9) in preparation for leaching that deposit in situ. Old Reliable is now producing about 20,000 lb of cement copper daily.

## Brazil Approves Report on Carajas Fe Study

Amazonia Mineracao S. A., whose shareholders are Cia. Vale de Rio Doce and Cia. Meridional de Mineracao, a Brazilian subsidiary of U.S. Steel Corp., announced that the National Department of Mineral Production (Ministry of Mines and Energy) has approved the report of the detailed prospecting carried out at Serra Dos Carajas in the east-central portion of the State of Para.

The Carajas prospect is potentially one of the world's largest iron ore deposits. U.S. Steel has a 49% interest in the project. Exploration and drilling reports indicate that the average iron content of the ores at this location is approximately 67% and the proven reserves are in excess of 1.5 billion metric tons. Additionally, there are indicated and presumed reserves substantially beyond this amount.

As a consequence of the study, advance engineering will proceed to further outline the possibility of mining and processing the iron ore. Depending upon the outcome of these studies, the first iron ore from Carajas could be available as early as 1978.

*continued on page 12*