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MINES BRANCH INVESTIGATION REPORT IR 61-129

**GOLD EXTRACTION TEST WORK CARRIED
OUT FOR HALLNOR MINES LTD.,
PAMOUR, ONT.**

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by

W. A. GOW, R. M. ENNIS & F. J. KELLY

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EXTRACTION METALLURGY DIVISION

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Mines Branch Investigation Report IR 61-129

**GOLD EXTRACTION TEST WORK CARRIED OUT FOR HALLNOR MINES LTD.,
PAMOUR, ONT.**

by

W.A. Gow^{*}, R.M. Ennis^{}, F.J. Kelly^{***}**

SUMMARY OF RESULTS

The decreased gold extraction and increased reagent consumption experienced during the early part of 1961 were probably due mainly to a relatively high pyrrhotite content in the mill feed along with high lime levels in the plant solutions.

Some of the cyanide consumption is due to the reaction of cyanide with chalcopyrite.

Improved control of the lime concentration in the plant solution throughout the circuit and particularly in the re-grind mill to a minimum concentration consistent with good thickener operation would probably result in lower reagent consumption and less solution fouling. The lime level for minimum pyrrhotite dissolution should not be higher than 0.1 to 0.2 lb/ton, corresponding to a pH of 11.2. If the thickener operation requires lime levels appreciably higher than this, the use of a non-alkaline flocculant may be desirable.

Additional air in some parts of the circuit may be beneficial.

A reduction in the amount of lead oxide added may be possible.

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INTRODUCTION

In April 1961, Mr. D. Bridger, manager of Hallnor Mines Ltd., Pamour, Ont., wrote to the Mines Branch requesting that the Extraction Metallurgy Division undertake an investigation of the cyanidation of their ore. Mr. Bridger, in his letter of April 20, 1961, stated that they were experiencing difficulty in maintaining gold extraction at the level of 98% which had been customary up to that time. He also pointed out that cyanide and lime consumptions were increasing as the tendency toward lower extraction became apparent. The Hallnor staff thought that all of these problems might be due to an increase in the pyrrhotite content of mine ore taken from the lower levels.

Mr. Bridger's request was that the Extraction Metallurgy Division carry out a test program designed to determine the reasons for these difficulties in the treatment of their ore and to suggest possible solutions.

In response to Mr. Bridger's request, a team of Mines Branch personnel including Messrs. W.A. Gow, F.J. Kelly and R.M. Ennis of the Extraction Metallurgy Division and Mr. G.O. Hayslip of the Mineral Processing Division visited the Hallnor Mill on May 29-30, 1961. The purpose of this visit was to collect data and samples and carry out analyses on plant products in the mill. It was hoped that when all this information had been collected, it would then be possible

either to make some suggestions as to where changes in the mill operations might be beneficial or to decide from the facts then available what additional data would be necessary and whether they could be obtained from a laboratory test program.

This report gives the results of the field and laboratory work carried out by the Extraction Metallurgy Division staff including comments on the mill operations as it was at the end of May 1961.

PROCEDURE

During the visit to the mill, data were collected by carrying out on-the-spot analyses of plant solutions; by collecting samples for more detailed analyses and study at the Mines Branch laboratories; and by discussing the operations with the mill staff.

Analyses carried out in the plant included analyses of solutions from various points in the circuit for NaCN, both free and total, CaO, CNS, reducing power, pH and oxygen content. Solid samples were assayed for gold. The pregnant solution was checked for solids content, before and after clarification, by means of a Hellige Turbidimeter.

The samples taken for additional studies and analyses at the Mines Branch laboratory, and the investigational work done on them, were as follows:

- (a) A sample picked from the mill feed to represent typical mineralization, for mineralogical study for determination of the metallic minerals present and the occurrence and association of the gold.
- (b) A sample of representative mill feed for spectrographic and chemical analyses.
- (c) A sample of flotation concentrate (cyanidation feed) for spectrographic and chemical analyses.
- (d) A sample of cyanide tailings for sizing analysis, and for determination of the distribution of the gold in the various sizes.
- (e) Samples of pregnant and barren solutions for chemical analyses and gold leaf dissolution tests.
- (f) A sample of gold precipitate for chemical analyses.

RESULTS

A mineralogical examination of nine small hand-picked specimens of mineralized ore, and six pieces of 7/8" drill core, all chosen as representative of the mineralization of the mill feed, was carried out. The mineralogy was reported⁽¹⁾ as follows:

"The metallic minerals present are mainly pyrite and pyrrhotite, with smaller amounts of arsenopyrite and chalcopyrite, and trace amounts of native gold, sphalerite and rutile. The gold occurs as small irregular grains mainly within pyrite, but it was also observed in tourmaline, in pyrrhotite and in gangue minerals. The largest grain of gold observed is present in pyrrhotite (80 x 40 μ); in pyrite the grains range in size from less than 1 x 1 μ to approximately 20 x 32 μ .

Gangue minerals present include mainly quartz, along with some calcite, brown acicular tourmaline, fine-grained white to green mica (sericite), and chlorite."

The results of chemical analyses at the plant and at the Mines Branch laboratories on solutions and solids taken from 15 points in the plant circuit are shown in Tables 1 and 2. The points from which the samples were taken are shown on the flow diagram (Figure 1).

The regrind mill discharge (Sample 3) and the No. 1 Agitator (Sample 4) were sampled two, and five times, respectively, to check for variations in the lime and cyanide concentration at these points. The results are shown in Table 3.

The results of the spectrographic semi-quantitative analyses on mill feed, flotation concentrate and residue from evaporation of the pregnant solution are shown in Table 4.

Screen and infralyzer sizing analyses were done on the cyanide tailings with the results shown in Table 5.

Table 6 is a tabulation of average operating data from the mill's 1960 records. In addition to the data shown in Table 6 other significant information came out in discussions with the mill staff. For example, it was stated that the pyrrhotite content of the flotation concentrate, as determined by the "Noranda method", is higher now than in the past and has gone as high as 28% pyrrhotite; also that a gradual cut-back in the lime concentration carried in the mill solutions resulted in some improvement in the overall operation.

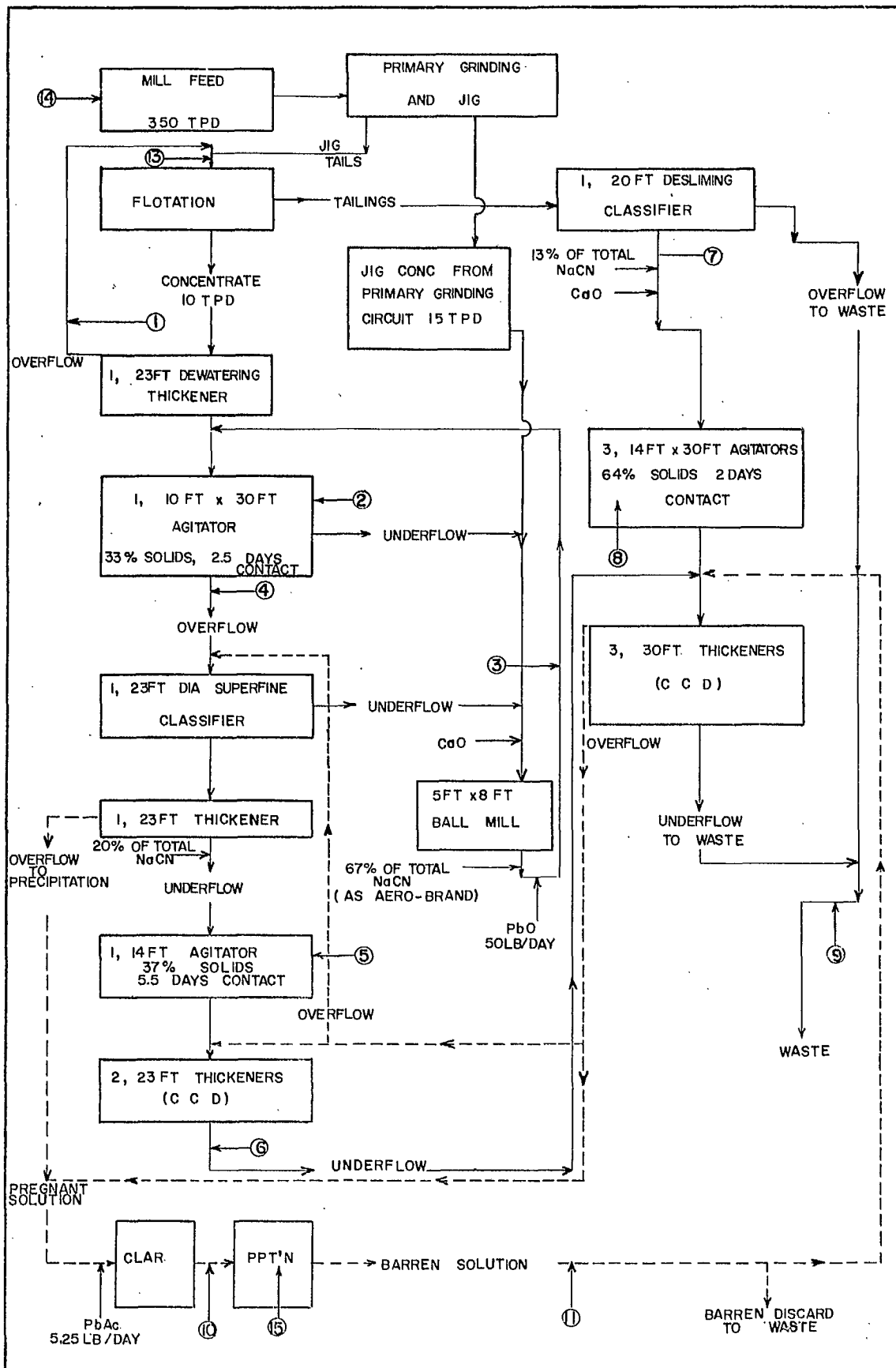


FIGURE 1 FLOW DIAGRAM OF HALLNOR MILL

TABLE I

Analyses of Plant Solutions

Sample No.	Sample Point	pH	Reducing Power	Oxygen (% of Saturation)	NaCN Total (lb/ton)	NaCN Free (lb/ton)	CaO (lb/ton)	CNS (g/l)	Au Assay of Solids (oz/ton)	Gold Leaf Dissolution Time (min)	Remarks
1.	O'flow from Concentrate Dewatering Thickener	8.2	--							40*	Au assay of solids in underflow -4.44 oz/ton.
2.	Dip from No.1 Agitator	11.5	1555	100		0.28	0.32	1.53	2.85	195	Underflow from No.1 Agitator --30% O ₂ saturated.
3.	Regrind Mill Discharge	12.2	1260	0	0.86	0.12	1.2		3.99		
4.	No.1 Agitator O'flow	11.9	1430		1.07	0.44	0.56	1.55			
5.	No.2 Agitator	11.6	1400	73		0.26	0.32	1.57			
6.	No.3, 23 ft Thickener Underflow	11.7	570	0		0.36	0.44		0.025		
7.	20 ft Desliming Classifier Underflow	9.1	28	100		0.05	0	0.03	0.0125	30**	
8.	Last Agitator on Flotation Tails	12.2	56	69		0.64	0.76	0.05	0.008	120	
9.	Combined Tailings	9.4	120			0.10	0		0.008		
10.	Clarified Pregnant Solution	12.2	700		0.95	0.53	0.76	0.68		270	Unclarified preg cont 50 ppm of solids. Clarified preg cont 4 ppm of solids.
11.	Barren Solution	12.2	680		0.93	0.54	0.76	0.69		270	
13.	Flotation Feed								0.20		
14.	Mill Feed								0.47		
15.	Gold Precipitate (Analyses on Table 2)										

* 0.5 lb NaCN/ton sol added

** NaOH to pH 11.2 NaCN to 0.55 lb/ton added



*Dr Fortaw - P
note & file*

Y. O.F. per JGH

FILE NO

DEPARTMENT
OF
MINES AND TECHNICAL SURVEYS
MINES BRANCH

Extraction Metallurgy
Division

300 Lebreton St.,
Ottawa, Ontario,
March 13, 1962.

*e.T.G.F.
File No. 42 A/11-1(0-2)*

Dr. J.M. Harrison,
Director,
Geological Survey of Canada,
601 Booth St.,
OTTAWA, Ontario.



Dear Dr. Harrison:

Re: Mines Branch Investigation Report IR 61-129

"Gold Extraction Test Work Carried out for Hallnor Mines Ltd., Pamour, Ont."

The above mentioned report was mailed out on or about March 5, 1962 and, according to our records, you received a copy.

We have noted an error on Page 7, Table 2, in the column headed "Gold Precipitate". The copper, nickel and zinc analyses should read 2.64%, 0.15% and 14.9% respectively instead of the values now shown in the Table. We would appreciate it if you would amend your copy of this report to include the above correct values.

Yours very truly,

W.A. Gow,
Senior Development Officer,
Hydrometallurgy Subdivision.



e.T.G.F. March 15, 1962 - JGH

MAR 15 1962

TABLE 2

Chemical Analyses of Some Plant Products

(Solid samples, %; solutions, g/l)

Assayed for	Mill Feed	Flotation Conc.	Pregnant Solution	Barren Solution	Gold Precipitate
Cu	< 0.01	0.37	0.063	0.073	18.82
Ni	< 0.005	0.075	0.014	0.014	0.03
Co			< 0.01	< 0.01	
Fe					0.53
S (total)		15.1	0.48	0.48	3.65
As	0.026	0.33			
C (as carb)	2.87	3.05			
C (total)	3.05	8.80			
C (carboni-ferous)	0.18	5.75			
S/SO ₄			0.05	0.06	0.60
S (sulphide)					2.70
Pyrrhotite		17.5			
SiO ₂					11.5
Zn					20.1
Pb					13.9
Au, oz/ton	0.47	4.44			
Ag, oz/ton	0.05	0.43			
NaCN (free) lb/ton			0.56	0.56	
NaCN (total) lb/ton			0.90	0.92	
CaO, lb/ton			0.68	0.80	
CNS			0.74	0.72	
Fe(CN) ₆			0.007	< 0.005	
pH			11.8	11.9	
R. P.			764	720	

64%

0.15%

14.9%

TABLE 3
Variation in Plant Solution Analyses

Sample No.	Sample Point	Time	NaCN (lb/ton)	CaO (lb/ton)	pH
3	Regrind Mill Discharge	May 29, 11:30 AM	0.12	1.2	12.2
		May 30, 9:00 AM	0.20	0.86	12.2
4	No.1 Agitator	May 29, 11:30 AM	0.28	0.32	11.5
		May 30, 9:00 AM	0.39	0.72	12.0
		May 30, 2:30 PM	0.44	0.56	11.9
		May 30, 3:00 PM	0.38	0.72	12.0
		May 30, 3:30 PM	0.40	0.80	12.1

TABLE 4
Spectrographic Semi-Quantitative Analyses of Some Plant Products, (%)

	Mill Feed	Flotation Concentrate	Solids from Evaporation of Pregnant Solution
Ba	0.03	0.03	
Si	15 PC	10 PC	0.2
Mn	0.1	0.2	
Al	2	2	0.01
Fe	3	13	0.02
Mg	3	4	0.006
Cr	0.02	0.02	
V	0.02	0.02	
As	ND	0.3	
Zn	ND	0.07	0.1
Mo	0.01	ND	
Ca	7	7	PC
Cu	0.03	0.3	2
Zr	Tr	Tr	
Na	0.5	0.3	PC
Ti	0.2	0.3	
Ni	0.01	0.1	0.2
Co	0.006	0.08	0.03
B	Tr	Tr	ND
Pb			0.08
Ag			0.01

PC Principal constituent

ND Not detected

Tr Trace

TABLE 5

Sizing Analyses of Hallnor Cyanide Residue Showing Gold Distribution

Size	Wt (%)	Au Assay (oz/ton)	Au Distribution (%)
+200 Mesh	1.1] 0.17	39.2
- 200 M + 56 μ	0.1		
- 56 μ + 40 μ	0.9] 0.06	25.7
- 40 μ + 28 μ	4.3		
- 28 μ + 20 μ	9.1] 0.0425	35.1
- 20 μ + 14 μ	14.0		
- 14 μ + 10 μ	14.9		
- 10 μ	55.6		
	100.0	0.067	100.0

TABLE 6

Average Data for 1960 Operation

(from mill records)

Reagent Consumption (lb/ton mill feed)

Aero-Brand Black Cyanide	1.748
Sodium Cyanide	0.546
Total equivalent Sodium Cyanide	1.385
CaO	5.76
Zn dust	0.036
Lead Acetate	0.015
Lead Oxide	50 lb/day

Other Data

Daily tonnage	329
Gold head assay, oz/ton	0.398
Ratio of concentration in flotation	33.24:1
Overall recovery, %	97.96
Solution to precipitation, tons/day	500
Barren to waste	nil*

* At time of plant visit in May 1961, 130 tons per day of barren was being discarded.

DISCUSSION OF RESULTS

A consideration of the data, shown in the Tables in this report, in the light of previous experience both in operating plants and in laboratory test work on other ores at the Mines Branch, suggests that some of the difficulties experienced in the operation of the plant in the early part of 1961 were due to the presence of increasing amounts of pyrrhotite in the ore along with too high a lime concentration in the plant solutions. Previous test work has shown that when pyrrhotite is agitated in a lime-cyanide solution, the lime and cyanide consumption and the degree of solution fouling increases rapidly as the pH increases above 11.2. A pH value of 11.2 is equivalent to 0.1 to 0.2 lb CaO/ton of solution depending on what other salts are present in the plant solutions.

Since the sample of flotation concentrate (cyanide feed) taken by the Mines Branch contained 17.5% pyrrhotite (Table 2), and statements by the mill staff indicated that the product had contained up to 28% pyrrhotite, it seems likely that the higher reagent consumption and lower extraction recently experienced is due to the reaction of pyrrhotite with the lime at the relatively high levels of lime being carried in the mill circuit. Table 1 shows that the solutions in the mill contain lime concentrations ranging from 0.32 to 1.2 lb CaO/ton of solution. These values are all higher than the lime concentration, of 0.1 to 0.2 lb/ton, above which the pyrrhotite reaction accelerates. Consequently, there is a good chance that lowering the lime concentration of the solution throughout the mill circuit, and particularly in the regrind circuit,

could result in lower reagent consumption and increased extraction. It is appreciated that lime is being used to settle the thickeners in the circuit, and, therefore, the cut in the lime concentration would have to be consistent with good thickener operation. However, it might be profitable to use a synthetic flocculant for thickening so that the lime requirement would not be controlled by the thickening operations.

It was noted that the lime feeder in the regrind circuit was already travelling at its slowest speed. Consequently, if the lime level in the regrind mill is to be reduced, mechanical modification in this lime feeder would be necessary.

Table 1 also shows the degree of oxygen saturation in some of the mill pulps. The zero oxygen content in the regrind mill discharge and thickener underflow is consistent with high pyrrhotite and high lime conditions since the pyrrhotite-lime reaction consumes oxygen rapidly. The rather low (about 70%) oxygen content in the agitators is probably due also to the pyrrhotite content. Although it is not possible to say that there is an oxygen deficiency in the agitators (since the minimum oxygen requirement is not known) it might be desirable to increase the air to the agitators for two weeks or so to see if any improvement results.

The gold leaf dissolution tests showed that the pregnant and barren solutions were the least active gold solvents of the plant solutions tested, each requiring 4 1/2 hours to dissolve the test leaf. This figure

in itself is not too significant. However, it may be used as a datum with which future gold leaf dissolution tests may be compared in order to indicate whether modification in the mill operation had resulted in less solution fouling. It can be said that a figure of 4 1/2 hours is comparatively low compared to those obtained in other mills where flotation concentrate is treated with cyanide.

The infrasizer analysis along with the gold distribution in the various sizes shown in Table 5 indicates that much of the gold lost in cyaniding the flotation concentrate is due to insufficient grinding. However, the amount of gold in the +20 μ sizes is not sufficient to warrant much additional grinding.

The mineralogy examination confirms much of the foregoing comment since it was found that pyrrhotite is a major metallic mineral constituent of the ore, and that gold occurs in grains down to 1 micron in cross-section.

The cyanide consumption in the mill can be almost entirely accounted for by the thiocyanate and heavy metal cyanide compounds in the plant solution. As has already been discussed, low lime will minimize the pyrrhotite reaction, the products of which will consume cyanide to produce thiocyanate and ferrocyanide. In addition it is believed that high lime and cyanide increase the solubility of chalcopyrite with its resultant increase in cyanide consumption as cupro-cyanide complexes. The presence of chalcopyrite was confirmed by the mineralogical study.

The amount of lead oxide being added to the regrind mill discharge (50 lb/day \cong 5 lb/ton conc) may be higher than necessary. Consequently, some savings in the reagent cost may be possible. Previous test work on a similar concentrate has shown that lead oxide in amounts over 1-2 lb/ton does not have an increasingly beneficial effect.

Although there was no suggestion by the mill staff that precipitation was causing difficulties, the analysis of the precipitate (Table 2) shows that the silica, lead and copper contents of the precipitate are on the high side. The copper may be reduced if lime levels in the plant are lowered, and the lead may be reduced if the lead oxide addition is lowered. The silica content may indicate that improvements in clarification are possible. This possibility is also indicated by the 4 ppm of solids present in the clarified pregnant solution, a solids content which could perhaps be reduced.

ACKNOWLEDGMENTS

The contribution of others on the Mines Branch staff to this investigation included the chemical analyses done by Mr. E. Kornelsen and his group, under the direction of Mr. J. C. Ingles, in the Extraction Metallurgy Division laboratories. The spectrographic analyses and the gold assaying work were done under the supervision of Dr. A. H. Gillieson and Mr. L. Lutes, respectively, of the Mineral Sciences Division. The authors also appreciate the assistance given by the Hallnor assay office staff who carried out some of the gold assaying work.

REFERENCE

1. S. Kaiman. Mineralogy of Gold Ore from Hallnor Mines Ltd., Pamour, Ont. Mines Branch Investigation Report IR 61-127, Department of Mines and Technical Surveys, Ottawa, (1961).

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