FEASIBILITY STUDY OF THE SAN FERNANDO OREBODY IN THE STATE OF SONORA MEXICO

By

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T 1461

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ABSTRACT

The purpose of this study is the investigation of the feasibility of the exploitation of the San Fernando orebody in the State of Sonora Mexico. The system of analysis consists of a series of cost estimates for mining and metallurgical equipment, as well as the estimates of ownership and operating costs of the whole mining-metallurgical operation. Some estimates are based on actual mining operations as they appear in literature, and other are the result of short calculations.

The principal goal of the problem is to calculate the discounted-cash-flow rate of return (D.C.F.R.O.R.), and to take it as a profitability index on the basis of a conservative rate of return of 20% for speculative enterprises.

The investigation finishes with a sensitivity analysis to take into account the effect of the different parameters of the D.C.F.R.O.R. calculations, that is, to see how errors in the different estimates can modify the final decision concerning the feasibility of the project.

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CONCLUSIONS

The discounted-cash-flow rate of return (D.C.F.R.O.R.) for the investment is very low (5.00%), and under these conditions the enterprise is highly risky, and not to be highly recommended. Only when the market for metals is better or when the scarcity of the metal becomes acute, more favorable conditions are to be found for the utilization of this project. The principal fact that aggravates the situation is that no other substances of value are present besides copper.

Exploitation at higher rates could probably make the operation more economical but the life of the mine would be so short that better total profits are questionable. The same thing can be said of exploiting the orebody at lower rates. At the present time it is difficult to find financiers willing to invest money with so small an expected rate of return, and with unfavorable variations in copper prices which can bring the operation to a standstill.

Other factors which have not been considered, are the problem of pollution of the river waters which are utilized downstream for irrigation purposes, and the possibility of flooding of the pit. This is possible because of the proximity of the river to the projected pit (Fig. 22). The control of these factors represents extra disbursements to make the project more infeasible.

INTRODUCTION

The San Fernando project is located in the municipality of Camuripa in the State of Sonora Mexico, on the western part of the Sierra Madre Occidental, and is bounded by the left bank of the Yaqui river.

The access to the project is by means of a non-paved road from Camuripa to the El Bienestar ranch. The total length of this road is 18.6 miles (30 km). From this point to the area of interest a new road 7.45 miles long (12 km) was constructed by Asarco Mexicana, S. A.

Camuripa, has direct comunication with the towns of Esperanza and Ciudad Obregon along a 74-mile (120 km) road which is in a well maintained condition.

Connection with the Pacific Railroad at Ciudad Obregon provides transportation. From Ciudad Obregon shipments of concentrates can be made to the different smelter plants in or out of the country.

GEOLOGY

The geological exploration of the area of San Fernando was made by geologists of the exploration department of Asarco Mexicana, S. A.. The geological examination reveals that the mineralization in the area is made up of primary copper sulphides related to a monzonitic intrusive. The principal lithological units in the area are well classified in two different groups as preminerals and postminerals, which classified in the geological report dated June 28, 1971, are identified as follows:

Preminerals

In the premineral formations recrystallized calcareous sediments, monzonitic porphyry, and tactites are taken into consideration.

Monzonitic Porphyry. In the area the intrusive outcrops with continuity over a zone of 44.5 acres (18 hectares); it continues to the north and west under the calcareous sediments.

According to field observations and drilling data obtained by the geologists of Asarco Mexicana, the conclusion has been reached that the intrusive is a lenticular sill wedged toward the east and southwest, and dipping to the northwest. Laterally, the porphyry varies in composition from monzonite to quartz-monzonite, and probably to granodiorite. The texture varies from porphyric to equigranular, and when the porphyry is greatly altered, it is recognized by the quartz eyes, by the sericitized micas, and by its thicker texture.

Besides the body of porphyry, some radial dykes are present to the east and northeast of the zone. In this last direction they penetrate under the river, passing to its north bank. According to the drilling data, these intrusive bodies are present also under the lower surface of the principal body. The maximum thickness registered for the monzonite body is approximately 525 ft (160 m).

Recrystallized Calcareous Sediments. These sediments are present throughout the region, and on this area they are exposed as products of erosion in the form of benches. These sediments were recrystallized by the intrusion of a monzonitic body, with an effect of slight folding. The result in the area of contact is an irregular-shaped intrusive.

Besides these limestone benches, bodies of calcareous breccia are present as products of the original limestones, and the cementing material is calcium carbonate derived from the limestones themselves.

Tactites. Under the name of tactites, two types of rock are grouped with different characteristics. Macroscopically they are difficult to distinguish one from another in some parts of the area because of the effects of weathering.

The first type of these rocks consists of calcium silicates (garnet, epidote, etc.), which are products of contact metasomatism generated by the intrusion of the porphyry in the limestones, and they are precisely located in the contact between these two rocks. Also with this type of metasomatic rock some hypogenic specular hematite is embedded.

The second type of rock is fine-grained with colors varying from light-gray and green, to dark-green and black. according to petrographic studies on core samples, these rocks seem to be of volcanic origin, and of a predominant andesitic composition. They present an intense fracturing in several directions, although a definite parallel pattern is distinguished on the eastern part of the area. This rock is also intruded by the monzonite, which is intercalated at depth.

Postminerals

Two types of lithological units are distinguished in this group: andesite dykes, and tuff.

Andesite Dykes. In this area a dyke of andesite of olive-green color was observed. The structure has an average thickness of 13.1 ft (4 m), striking N 10^{o}W and dipping 35^{o} to the southwest. The dyke cuts the second type of tactite and the monzonite on the east of the area.

Tuff. The rocks specified above overlie a group of intercalated volcanic tuff of andesitic and rhyolithic composition. It seems that the tuff and lavas were deposited when the erosion was acting intensely on the limestone. This can be inferred because the tuff and the lavas occur in contact overlying the calcareous sediments, the monzonitic intrusive, and the metasomatic rocks.

Hydrothermal Alteration

The predominant alteration is in the contact of the monzonitic body with the country rock, and consists of weak argilization and silicification. The alteration is weak; however, there are some parts in which silicification is moderate, and yet it is possible to find minor quantities of quartz-sericite. Also, almost throughout the area minor quantities of tourmaline are found.

Mineralization

The mineralization is represented by limonites principally of pyrite. These limonites are found locked in fissures, as well as disseminated. Only in the neighborhood of drill hole 5, some copper limonites are observed principally in fissures.

It is also possible to find fresh sulphides on road cuts where intense silicification is present. Films of chalcocite over pyrite are not rare.

One thing to note also is that in almost all the area copper carbonates are observed in the form of transported malachite and traces of azurite. This type of mineralization is the one which is exploited on a small scale in the north-west part of the area.

EXPLORATION

Information given in this section is also taken from the geological report of June 28, 1971 mentioned before.

The evaluation of the San Fernando project was made through two exploration steps. The first one, consisting of 16 drill holes driven with rotary equipment, gives a total drilled length of 4,560 ft (1,391 m). The second step, performed with diamond drilling equipment with core recovery, gives a total drilled length of 45,000 ft (13,697 m) distributed among 43 drill holes.

The construction of an access road was necessary for the drilling operation. Such a road runs for 12 km (7.45 miles) from the El Bienestar ranch to the area of interest. The construction of this road required pneumatic drills and the use of explosives.

The rotary drilling operation was carried out with success. The network of drill holes is presented in the surface map (Fig. 1). The drill hole assays are given in Table 1.

The configuration of the orebody can be followed through the vertical sections Fig. 2 through 6, and in the horizontal projections of benches Fig. 23 through 34.

On these last figures only the ore recovered with the projected pit is shown.

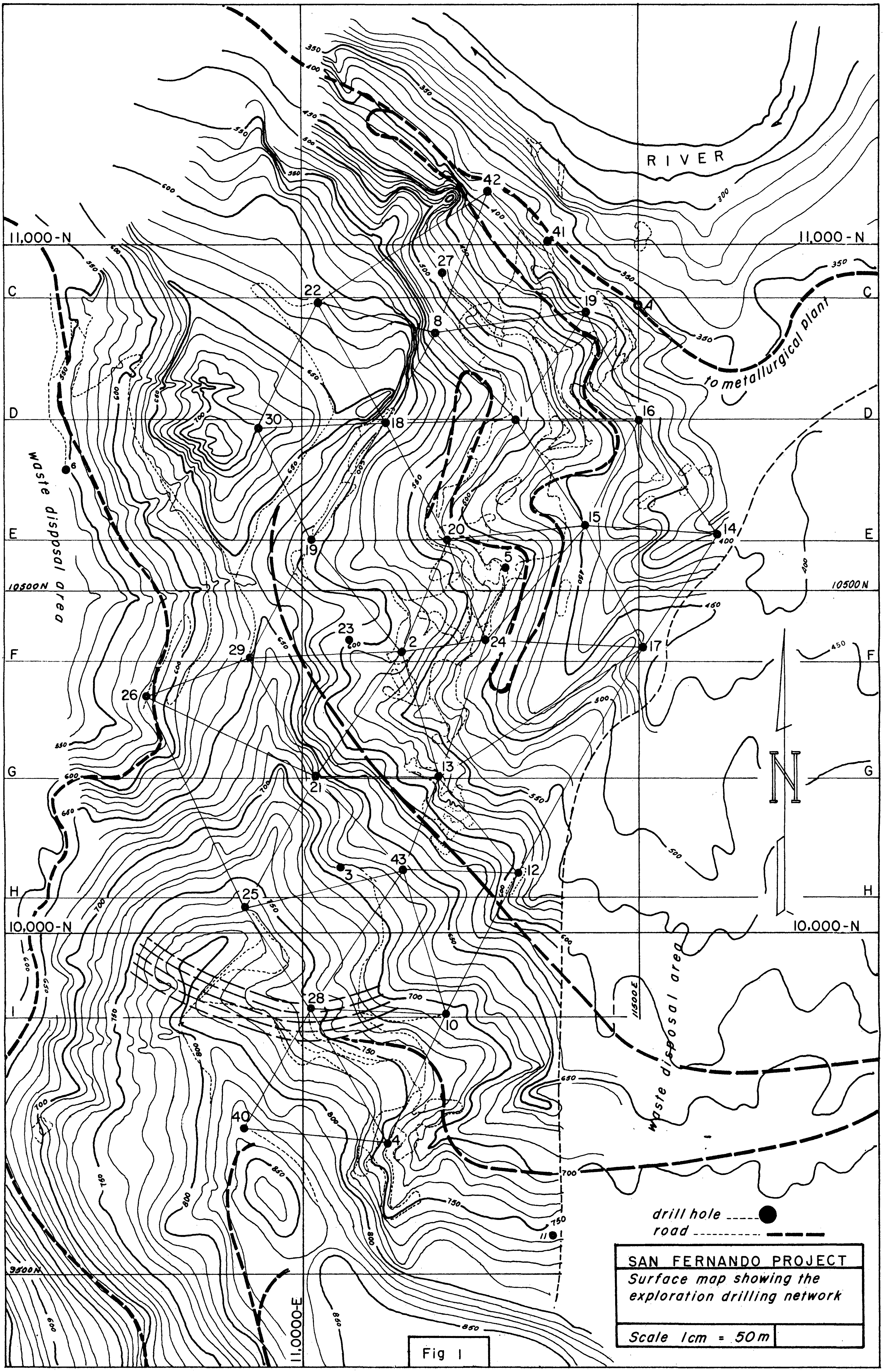
Table 1. Drilling Data

Drill Hole No.	Distance from Surface m	Distance from Surface ft	Interval m	Interval ft	% Cu Assay
1	15.45	50.6	22.70	74.5	1.02
	46.40	152.0	39.68	184.0	0.63
	108.88	357.0	23.43	76.9	0.73
	182.89	600.0	8.16	32.8	0.52
	165.80	544.0	6.75	22.1	0.73
2	60.54	198.5	53.28	174.8	0.59
	123.84	406.0	35.74	117.0	0.58
	264.90	870.0	8.58	28.2	0.74
	285.68	938.0	12.23	40.0	0.50
3	120.93	396.0	41.94	137.2	0.68
	200.33	657.0	39.56	129.5	0.68
	252.09	826.0	27.44	90.0	0.57
4	39.41	129.2	43.57	143.0	0.50
5	9.00	29.5	4.90	16.1	0.66
	17.92	58.7	59.14	193.5	1.16
8	103.00	338.0	2.82	9.3	0.65
	106.78	350.0	29.60	97.0	0.65
	192.10	630.0	9.99	32.7	0.59
	228.40	749.0	11.16	36.6	0.49
	291.41	955.0	14.96	49.0	0.63
9	72.13	236.0	6.10	20.0	0.63
	187.68	616.0	42.43	139.0	0.46
	279.58	916.0	25.34	83.0	0.48
	355.83	1,168.0	7.62	25.0	0.55
	371.08	1,216.0	6.10	20.0	0.68
10	98.52	323.0	84.77	278.0	0.30
12	3.05	10.0	15.42	50.5	0.62
	178.70	586.0	7.45	24.4	0.40
13	15.27	50.1	32.00	105.0	0.46
	65.58	215.0	39.64	130.0	0.30
	111.32	365.0	47.02	154.0	0.36
15	9.35	30.6	5.38	17.7	0.65
18	196.16	644.0	33.87	111.0	0.36
	242.63	796.0	12.08	39.6	0.53
	285.38	936.0	18.30	60.0	0.80
	313.19	1,025.0	15.68	51.5	0.54
	336.27	1,100.0	10.35	34.0	0.59
	395.29	1,295.0	16.64	54.6	0.48

Table 1. (cont'd.)

Drill Hole No.	Distance from Surface m	Distance from Surface ft	Interval m	Interval ft	Assay
19	31.20	102.5	56.72	181.0	0.60
	77.92	256.0	12.70	41.6	0.38
20	11.90	39.0	17.83	58.4	1.33
	40.90	134.0	125.84	413.0	0.68
	215.06	706.0	18.43	60.5	0.67
21	143.28	470.0	44.49	146.0	0.55
	180.00	590.0	25.00	82.0	0.72
	225.00	739.0	21.35	70.0	0.51
23	66.92	219.0	108.61	356.0	0.68
	183.40	601.0	29.20	95.6	0.73
	232.79	764.0	34.26	112.3	0.45
	279.25	916.0	38.89	127.5	0.48
24	1.52	5.0	11.22	36.9	0.48
27	52.12	170.8	21.79	71.4	0.59
	179.48	589.0	16.82	55.1	0.69
	216.53	711.0	39.78	130.5	0.49
	267.16	876.0	48.27	158.1	0.72
	327.25	1,072.0	6.69	21.9	0.51
28	194.95	640.0	19.22	63.0	0.65
	223.92	734.0	52.46	172.0	0.58
29	214.45	704.0	18.79	61.6	0.48
	245.61	805.0	24.77	81.4	0.56

Note: Missing holes in this table have not any value, but they have been taken into account on reserves calculation.



SAN FERNANDO PROJECT

Surface map showing the exploration drilling network

Scale 1cm = 50m

Fig 1

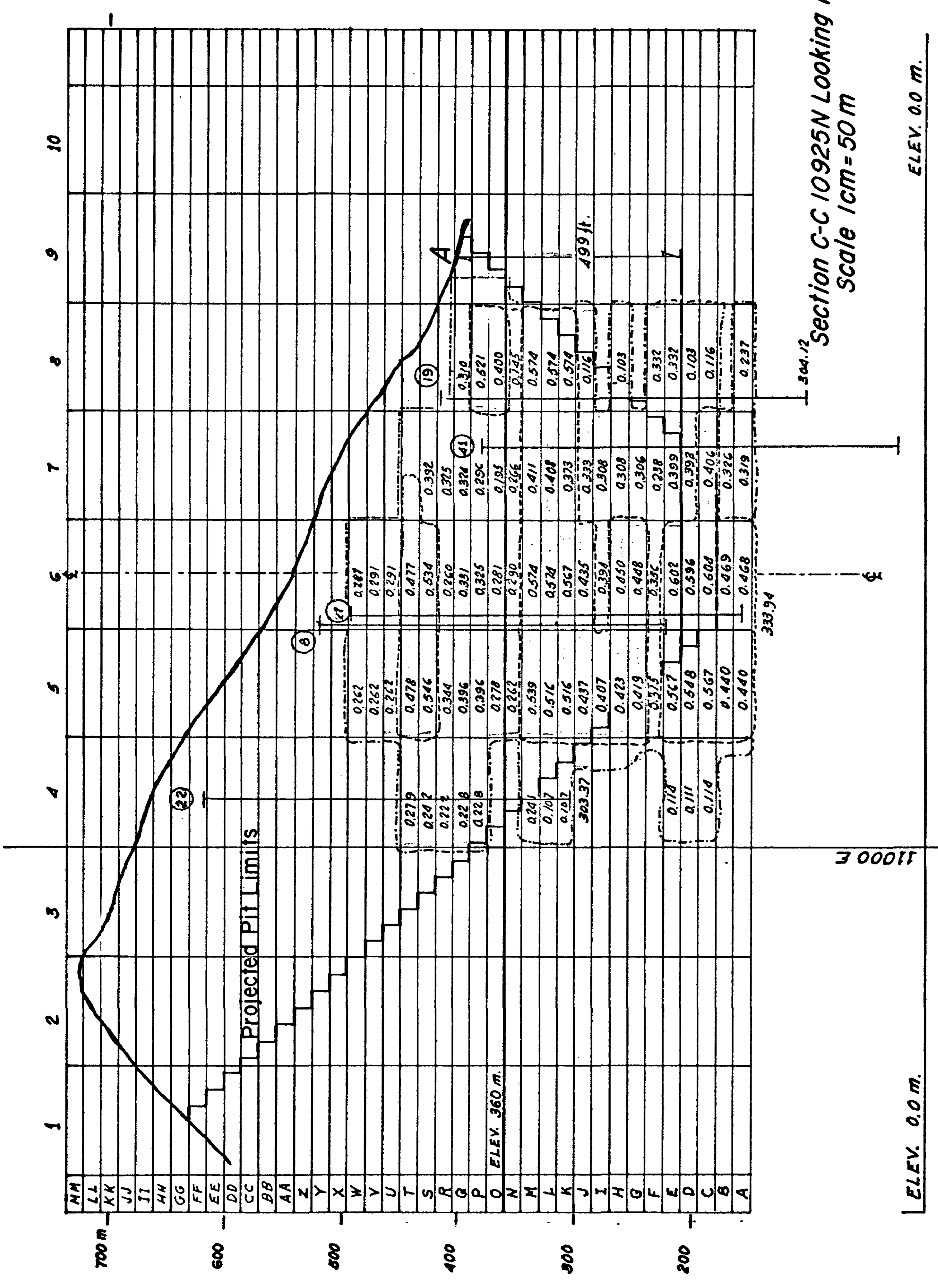

Section C-C 10925N Looking N
Scale 1cm=50m
Projected Pit Limits
ELEV. 360 m.
ELEV. 0.0 m.
ELEV. 0.0 m.
11000 E
499 ft.
304.12
303.37
333.94
700 m
600
500
400
300
200
1
2
3
4
5
6
7
8
9
10

Fig. 2

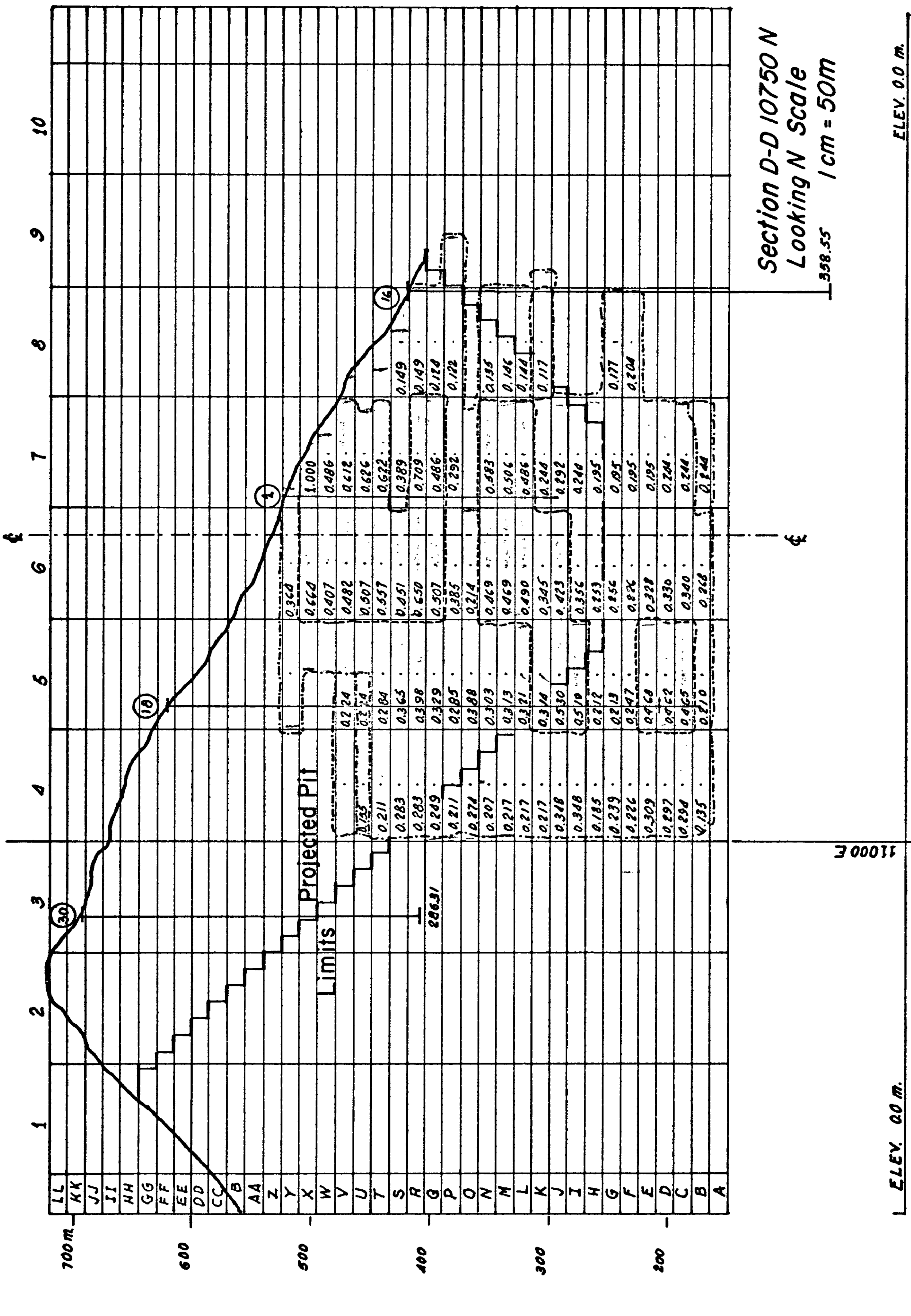

Section D-D 10750 N
Looking N Scale
1 cm = 50m
358.55
ELEV. 0.0 m.
Projected Pit
Limits
288.31
11000 E
ELEV. 0.0 m.

Fig. 3

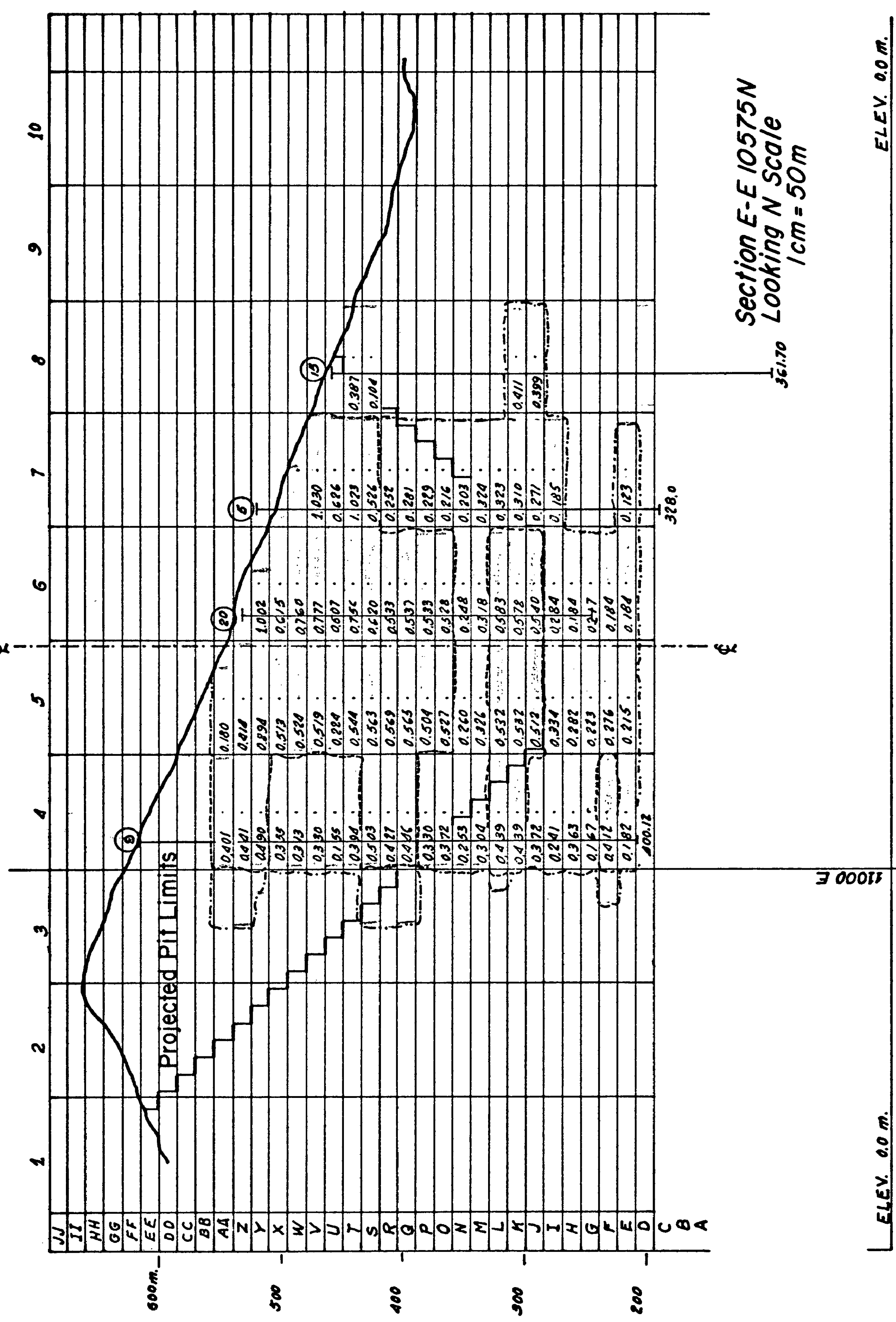

Section E-E 10575N
Looking N Scale
1cm = 50m
Projected Pit Limits
ELEV. 0.0 m.
11000 E

Fig. 4

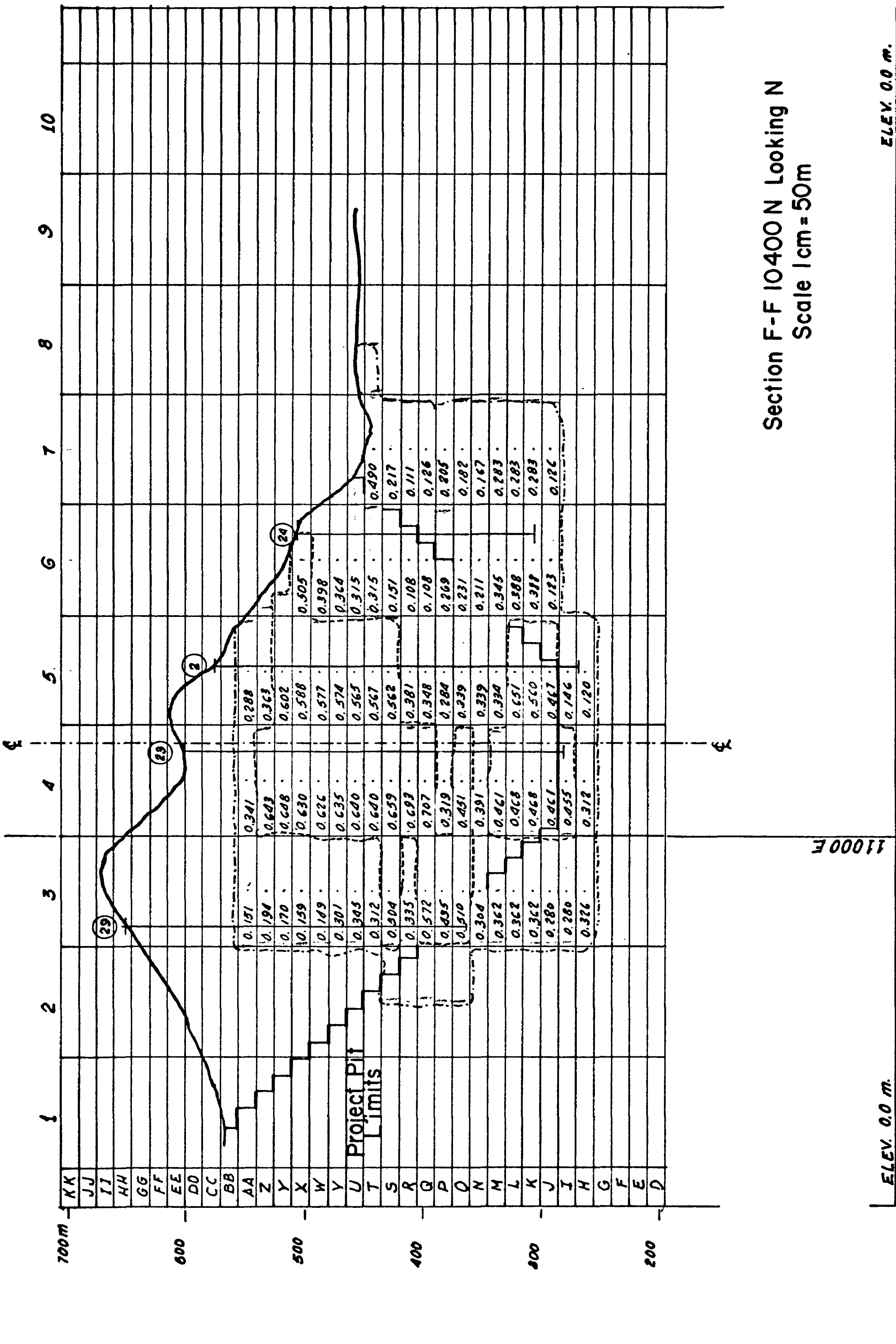

Section F-F 10400 N Looking N
Scale 1cm=50m
Project Pit Limits
ELEV. 0.0 m.
ELEV. 0.0 m.
11000E
700 m
600
500
400
300
200

Fig. 5

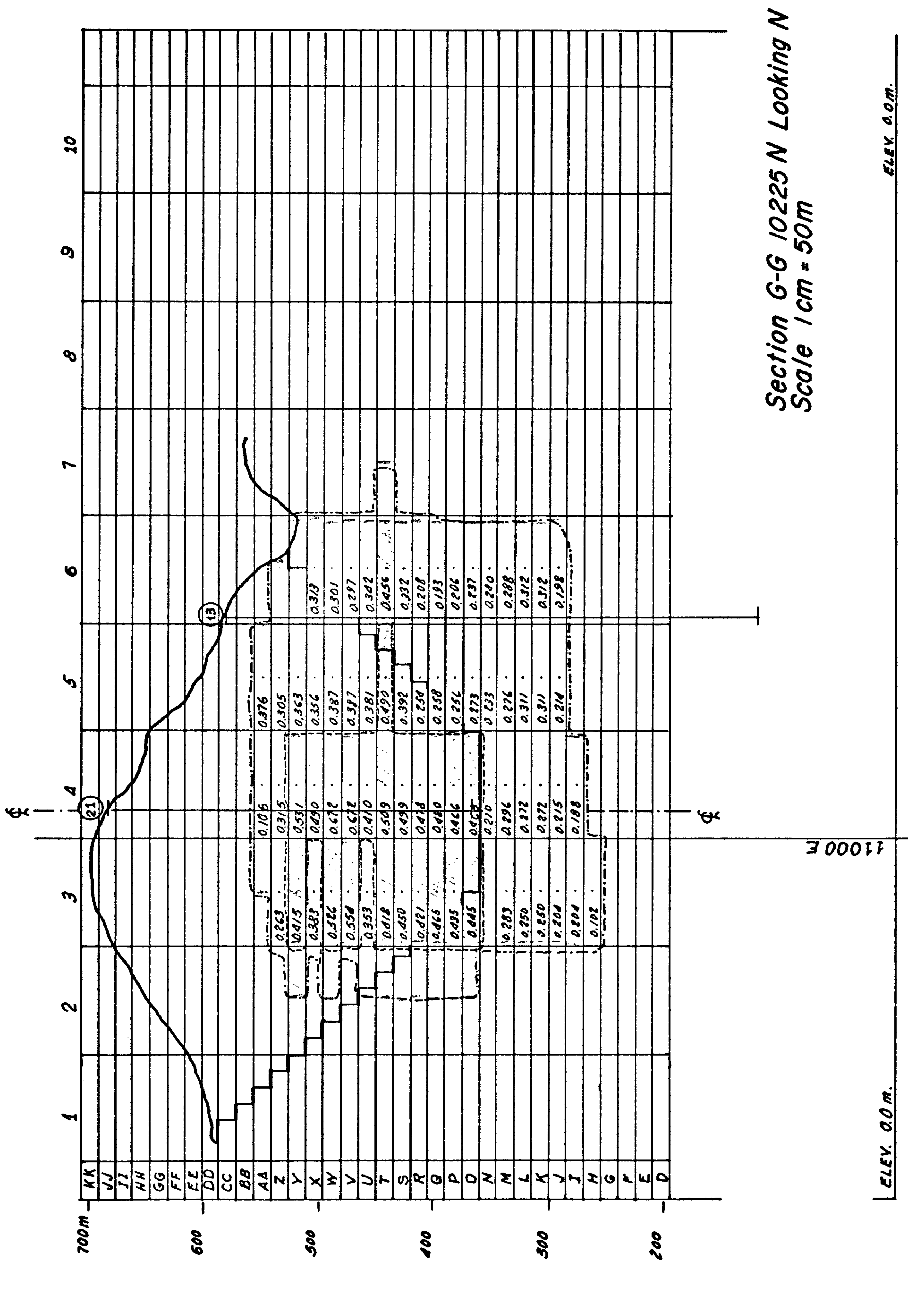

Section G-G 10225 N Looking N
Scale 1cm = 50m
ELEV. 0.0m.
ELEV. 0.0m.
11000 E
700m
600
500
400
300
200
1
2
3
4
5
6
7
8
9
10

Fig. 6

RESERVES CALCULATION

Appendix A carries the reserves calculations where all the calculations were recorded after being done with the use of a FORTRAN program; and the results are summarized in Table 2. The tonnages of ore within the range of 0.1 to 0.4% Cu were included because of the possibility of recovering values via leaching in the future.

Table 2. Summary of Reserves Calculation

Bench	Copper Tons from ore with grade <0.40%	Ore Tons with grade <0.40%	Copper Tons from ore with grade >0.40%	Ore Tons with grade >0.40%
AA	15,210	5,653,248	12,816	2,609,191
Z	19,035	6,522,979	17,797	3,478,922
Y	14,449	4,783,518	39,259	6,522,979
X	18,728	6,088,113	36,852	6,522,979
W	22,472	7,392,709	35,436	6,522,979
V	24,564	7,827,574	46,795	7,827,574
U	27,950	9,132,170	45,810	7,827,574
T	21,179	6,957,844	60,091	10,871,631
S	24,596	9,132,170	50,268	9,567,035
R	33,512	12,611,092	38,783	6,957,844
Q	30,281	11,741,361	37,194	6,957,844
P	33,971	12,615,501	27,534	5,653,248
O	28,348	11,306,496	20,833	4,348,652
N	37,823	15,655,148	6,730	1,304,596
M	35,512	13,045,957	23,913	4,783,518
L	21,964	9,132,170	44,089	8,697,305
K	26,642	10,436,766	36,536	7,392,709
J	24,979	10,001,900	27,291	5,653,248
I	25,742	10,001,900	10,587	2,174,326
H	22,484	9,567,035	5,658	1,304,596
G	15,516	6,957,844	5,605	1,304,596
F	20,754	8,697,305	3,697	869,730

Table 2. (cont'd.)

Bench	Copper Tons from ore with grade <0.40%	Ore Tons with grade <0.40%	Copper Tons from ore with grade >0.40%	Ore Tons with grade >0.40%
E	18,207	7,392,709	12,063	2,174,326
D	12,522	5,653,248	11,574	2,174,326
C	10,187	4,783,518	13,863	2,609,191
B	9,172	4,783,518	7,698	1,739,461
A	8,659	3,478,922	5,831	1,304,045
Totals	604,458	231,348,315	685,603	129,154,425

Tonnage Factor

On the basis of both an ore specific gravity of 2.63 as determined by Asarco Mexicana, and moisture content of 2.5% the tonnage factor is calculated as follows:

Moisture	2.5%
Specific gravity	2.63

$$\text{Pounds of ore per cu-ft} = 2.63 \times 62.5 \text{ lb/cu ft} \times (1.000 - 0.025 \text{ moisture}) = 169 \text{ lb/ft}^3$$

$$\text{Tonnage factor (T.F.)} = \frac{2,000 \text{ lb/ton}}{169 \text{ lb/cu-ft}} = 11.8 \text{ cu-ft/ton}$$

Ore Recovery

As may be expected, not all the ore available in the orebody can be extracted during the exploitation. Under these conditions a plan needs to be outlined in respect to the amount of ore that can be taken from the pit, without increasing the amount of waste removal beyond the point of profit cutoff. This means that an optimum pit exists where maximum profitability can be reached from the operation. Johnson (1971), and Lerchs and Grossmann (1965).

The preliminary design of pit for valuation purposes was made on the basis of maximum recovery of ore, assuming a final pit slope of 45°. For each vertical section (Fig. 2 through 6) the center line of the orebody was determined by taking moments of grade for each block with respect to the reference line 11,000-E in the same figures. The final walls of the pit were determined by making successive trials calculations of the waste/ore ratio by making the 45° pit walls to open or close in relation to the center line of the pit. The final results are plotted in the corresponding sections. The deepest point of the pit stands for section C. Measured in relation to point A in the same figure, this distance accounts for 499 ft (152 m), which is well within the capabilities of a truck-hauling system.

The preliminary design of the pit is shown in Fig. 22, and the configuration of benches is now available. In these conditions the tonnages of recoverable ore are well defined. The configuration of production benches is shown in Fig. 23 through 34, and those for stripping are shown in Fig. 8 to 21.

Table 3 summarizes the available and recoverable volumes of ore (above and below 0.40% Cu), the percentage of recovery of ore above 0.40% Cu per bench, and the recoverable tonnages of both ores. In this table the column headed as %-Recovery, gives the percentage of ore above 0.40% Cu which can be recovered from the given bench. The first entry of this column is obtained by dividing 21,175,680 recoverable tons by 31,763,520 tons available in the orebody. The overall percentage of recovery for the orebody becomes 82.9%, and this figure is given in the lowest part of the column.

Table 3. Summary of Available and Recoverable Ore

Bench	Ore Recovered Through Projected Pit: Ore above 0.4% cu-ft	Ore Recovered Through Projected Pit: Ore below 0.4% cu-ft	Total Ore in Orebody: Ore above 0.4% cu-ft	Total Ore in Orebody: Ore below 0.4% cu-ft	% Recovery	Recoverable Ore: Volume/T.F. Tons+0.4%	Recoverable Ore: Volume/T.F. Tons-0.4%
AA	21,175,680	68,291,568	31,763,520	68,820,960	66.8	1,794,550	5,787,423
Z	27,528,384	68,291,568	42,351,360	79,408,800	65.0	2,332,915	5,787,423
Y	70,409,136	38,101,464	79,408,800	58,233,120	89.0	5,966,879	3,228,939
X	62,997,648	51,668,659	79,408,800	74,114,880	79.4	5,338,786	4,378,702
W	74,644,272	74,644,272	79,408,800	94,096,640	94.0	6,325,789	6,325,789
V	85,232,112	82,055,760	95,290,560	95,290,560	89.6	7,223,064	6,953,881
U	82,055,760	76,232,448	95,290,560	111,172,320	86.3	6,953,881	6,640,380
T	112,231,104	56,115,552	132,348,000	84,702,720	84.6	9,511,115	4,755,557
S	111,172,320	61,144,776	116,466,240	111,172,320	94.2	9,421,387	5,181,763
R	74,114,880	94,231,776	84,702,720	153,523,680	86.9	6,280,925	7,985,747
Q	79,408,800	75,967,752	84,702,720	142,935,840	93.8	6,729,562	6,437,948
P	60,880,080	87,349,680	68,820,960	153,523,680	89.0	5,159,331	7,402,519
O	47,962,915	74,114,880	52,939,200	137,641,920	91.4	4,064,656	6,280,925
N	15,246,490	100,055,088	15,881,760	190,581,120	96.2	1,292,076	8,479,248
M	54,527,376	52,939,200	58,233,120	158,817,600	93.4	4,620,966	4,486,375
L	79,144,104	13,324,800	105,878,400	111,172,320	74.7	6,707,130	1,121,594
K	52,939,200	26,469,600	89,996,640	127,054,080	59.4	4,486,375	2,243,187
J	62,732,952	14,875,915	68,820,960	121,760,160	91.3	5,316,354	1,260,671
I	13,181,861	22,605,038	26,469,600	121,760,160	50.0	1,117,107	1,915,682
H	13,552,435	13,552,435	15,881,760	116,466,240	85.4	1,148,512	1,148,512
G	11,540,746	9,581,995	15,881,760	82,702,720	72.8	978,030	812,034
F	-	16,517,030	10,587,840	105,878,400	-	-	1,399,749
E	12,387,773	5,505,677	26,469,600	89,996,640	47.1	1,049,812	466,583
Totals	1225,066,028	1183,546,943	1477,003,680	2586,726,880	82.9	103,819,202	100,300,000

In resummary, the ore-reserves calculations are as follows:

Total tons of ore above 0.40% Cu available in the orebody	129,154,425
Total tons of ore below 0.40% Cu available in the orebody	231,348,315
Total recoverable tons above 0.40% Cu . . .	103,819,202
Total recoverable tons below 0.40% " . . .	100,300,633
Blocks of recoverable ore above 0.40% Cu. .	277
Sum of grades	146.180
Average grade = 146.180/277 = ...	0.53%
Blocks of recoverable ore below 0.40% Cu. .	493
Sum of grades	130.916
Average grade = 130.916/493 = ...	0.27%
Overall combined grade = $= \frac{104 \times 0.53 + 100 \times 0.27}{104 + 100} =$ . . .	0.40%

For exploitation purposes, the ore with average grade of 0.53% Cu, will be considered alone without any combination with the lower grade material. Ore below 0.40% Cu, is considered only as a possibility for recovering values through leaching, and in calculation of the waste/ore ratio is taken as waste.

Waste/Ore Ratio

The waste considered in this section comes from three sources :

a) From the stripping operations
b) From the production operations
c) Ore below 0.40% Cu

Table 4 shows the distribution of waste in the different benches, and Fig. 8 through 34 show complementary information.

Table 4. Distribution of Waste

Waste from Stripping		Waste from Production			
Bench	Tons	Bench	Tons	Bench	Tons
OO	2,720,000	AA	8,675,924	M	0
NN	3,490,000	Z	8,675,924	L	0
MM	4,260,000	Y	7,401,900	K	0
LL	4,790,000	X	7,693,490	J	0
KK	5,800,000	W	5,472,920	I	0
JJ	7,570,000	V	4,755,160	H	0
II	8,240,000	U	3,777,212	G	0
HH	10,920,000	T	2,803,750	F	0
GG	16,061,222	S	2,099,448	E	0
FF	12,158,076	R	1,543,184		
EE	13,548,852	Q	1,543,184		
DD	14,625,582	P	1,260,566		
CC	16,509,859	O	1,955,896		
BB	13,058,186	N	977,948		
Totals	133,751,777		58,636,506		

Waste from stripping	133,751,777	tons
Waste from production	58,636,506	"
Ore below 0.40% Cu	98,866,495	"
Total	291,254,778	"

$$\text{Waste/ore ratio} = \frac{291,254,778}{103,819,202} = 2.81$$

MINE VALUATION

The different items of the valuation of the project are covered one by one through this section. It is assumed that the rate of production for the mine is to be 30,000 tons of ore per day.

Life of Mine

At a rate of production of 30,000 tons of ore per day on a 6-days-a-week basis, the number of working days per year is approximately 300.

Yearly production = 300 days/year x x 30,000 tpd =	9,000,000 tons
Life of mine = $\frac{103,819,202 \text{ tons}}{9,000,000 \text{ tons/yr}}$ =	11.70 years
Stripping period as calculated later	3.43 "
Total operating period	15.13 years

It can be said that 12 years is the life of the mine, after a 4-year period of stripping. In this way the complete operation is assumed to be finished in 16 years.

Metallurgy

From the metallurgical treatment of the ore, only two products can be obtained: copper concentrate, and tails. Up to now, no metallurgical tests on the ore have been done, and therefore metallurgical problems can not be foreseen at this stage. However, for calculation purposes, Table 5 gives the possible recoveries of metallurgical treatment.

Table 5. Metallurgical Statement

Item	Weight Tons	Analysis Cu %	Recovery % Cu
Feed	30,000	0.53	--
Cu Conc.	523	28	92
Tails	29,477	0.085	8

Concentrate Production

The production of copper concentrate per ton of ore as calculated below accounts for 34.8 lb.

$$\text{Cu Conc./ton of ore} = 0.53\%\ \text{Cu} \times 20\ \text{lb}/1\% \times$$

$$\times\ 0.92\ R \times \frac{183.5}{63.5} \times \frac{34.6\%\ \text{Cu}}{28\ \%\ \text{Cu}} = \quad 34.8\ \text{lb}$$

$$\text{Concentration ratio} = \frac{2{,}000\ \text{lb of ore/ton}}{34.8\ \text{lb Cu-conc./ton}} = \quad 57.5$$

Market and Transportation

The concentrates obtained from the metallurgical process can be transported by rail to the different markets. The principal market could be a smelting plant to be installed by Asarco Mexicana in Nacosari Sonora on the assumption that the final product can be absorbed in both the national and international markets.

From the metallurgical plant of San Fernando project to the nearest railroad station, a distance of 74 miles (120 km) have to be covered by a fleet of trucks, which can make regular transportation to a dock at the railroad station.

From calculations it is estimated that three trucks are sufficient to transport the 523 tons of daily copper concentrate. 50-ton trucks are considered in these calculations, running at an average speed of 30 miles per hour. Considering one spare unit for maintenance purposes the total units becomes 4.

The railroad distance to the smelter plant is approximately 372 miles (600 km), covered at an average cost of $ 0.016 per ton-mile charged by the national railroad.

Water Availability

Water is directly available from the river which is approximately 1,300 ft from the proposed metallurgical plant, and 98.5 ft lower in elevation, Fig. 35.

There is no~~t~~ restriction in water consumption, although it is estimated that three tons of water will be consumed per each ton of ore treated. Under these conditions, the total consumption is estimated to be 90,000 tons per day.

Power Availability

Energy for the pit can be provided by the Federal Commission of Electricity (Comision Federal de Electricidad), with the construction of a 74-mile transmission line between Ciudad Obregon and the mine.

The power requirements are estimated in Appendix B, and the distribution is as follows:

Metallurgical plant	9,270	kva
Mine	3,700	"
Townsite facilities	2,030	"
Total	15,000	"

The recommended transmission voltage is 115 kv, and consumption costs are established in tariff 11-N of the Federal Commission of Electricity. The following covers the transcription of the tariff referred to.

Comision Federal de Electricidad

Tariff 11-N

High Tension Service For Mines

1) Application

No high-tension service under 20 kw of demand will be contracted for mines and related metallurgical plants.

2) Monthly Charges

(2-a) The fixed charges independently of the electrical amounts to $12.00 pesos ($0.96) a month per each kilowatt of basic demand.

(2-b) The additional charges for energy consumption are as follows:

Cost	Description
$0.20 (0.0160)	Per each one of the first 90 kwhr corresponding to each kilowatt of basic demand
$0.16 (0.0128)	Per each of the following 180 kwhr corresponding to each kilowatt of basic demand.
$0.11 (0.0088)	Per each of the remaining kwhr. Calculations are made as before.

3) Contracted Demand

The contracted demand is fixed by the consumer when the contract for service is signed, The demand will neither be less than 20 kilowatts, nor less than 60% of the total connected load. Also, it can not be less than the capacity of the largest motor installed by the consumer.

If the maximum demand as measured in three consecutive months becomes larger than the actual contracted demand, this fact modifies the contracted demand to the average of the three maximum measured demands.

4) Measured Maximum Demand

The measured maximum demand will be determined each month by means of registering instruments. The measured maximum demand stands for the highest average load registered in a fifteen-minute interval on any day in the month.

5) Basic Demand

The basic demand consists of the kilowatts either of the actual contracted demand or the maximum demand in a given month, whichever is larger.

6) Guarantee Deposit

The guarantee deposit is four times the applicable fixed charges.

Cost Estimate for Substation and Related Accessories

The cost estimate for the substation and related accessories was provided by Mr. E. Romero of the Comision Federal de Electricidad. The electrical facility consists of the following parts:

a). One linking substation
b). One 115-kv transmission line
c). Principal mine substation
d). Equipment feeding substations

Linking Substation. This linking substation is necessary at Cd. Obregon for matching of the 115-kv transmission line to the power source.

The estimated cost of this installation is given below:

Estimated Cost of Linking Substation

Item	Pesos	Dollars
Metallic Structures	$100,000	$ 8,000
Power Oil Circuit Braker	750,000	60,000
Relays	180,000	14,400
Instruments Panel	30,000	2,400
Cable and Buses	10,000	800
Grounding Network	10,000	800
Labor	60,000	4,800
Foundations	20,000	1,600
Other	16,000	1,280
Totals	$1,176,000	$94,080

115-kv Transmission Line. The installation of this transmission line involves the use of ASCR 336.4 MCM cable.

The supporting structures are type H with concrete poles 49.2 ft (15 m) high. The average span is 755 ft (230 m) with two lighting protecting ground wires (AGAR 0.312 in.). Eight insulators are required in tangents and suspension structures, and 9 pieces in angles at endings.

The cost of transmission line per kilometer including material and labor amounts to $200,000 pesos ($25,700/mile).

Total cost transmission line =
74 miles x $25,700/mile = $ 1,901,800

Principal Mine Substation. The electrical diagram of Fig. 7 shows the basic components of the principal mine substation. The subsequent table summarizes the cost estimate of materials and labor. Numbers in this diagram correspond with the items of the table.

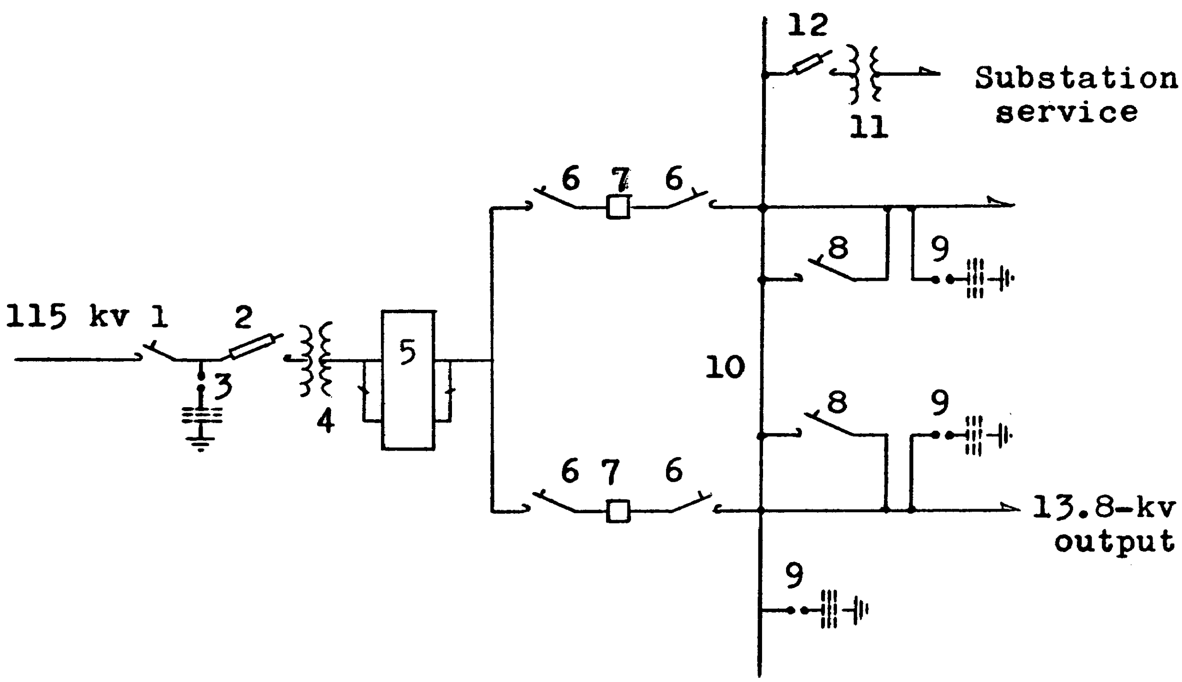


Fig. 7 Diagram of Principal Mine Substation.

Cost Estimate of a 15,000-kva Substation

Item	Quantity	Description	Cost Pesos	Dollar
1	1	Drawout bar 3 poles, 1 throw, 115 kv	$ 45,000	3,600
2	1 set	Fuses 115 kv, 600 amp	120,000	9,600
3	1 set	Lighting arresters	36,000	2,880
4	1	Transformer 15 mva*, 115/13.8 kv	900,000	72,000
5	1 set	Drawout bars, and measuring equipment (1 package)	20,000	1,600
6	12	Drawout bars 15 kv, 1 pole, 1 throw	20,000	1,600
7	2	Interruptors 500 mva, 1,800 amp	360,000	28,800
8	2 sets	Drawout bar 3 poles, 1 throw, 15 kv	40,000	3,200
9	9	Lighting arresters GP-DV-12	13,000	1,040
10	2 sets	Buses and bars mounted on metallic structure	15,000	1,200
11	1	Transformer 15 kva for service of substation	7,000	560
12	1 set	15-kv fuses	1,000	80
	General:			
		Relays	200,000	16,000
		High and low tension metallic structures	200,000	16,000
		Grounding network	10,000	800
		Lighting	20,000	1,600
		Fence	20,000	1,600
		Control cabin	40,000	3,200
		Insulators and ironwork	20,000	1,600
		Total Materials	2,087,000	166,960
		Salaries and wages	200,000	16,000
		Insurance and other fringe benefits $200,000 x 60%	120,000	9,600
		Other direct expenses	20,000	1,600
		Divisional expenses	60,000	4,800
		Indirect expenses control offices	60,000	4,800
		Total	$460,000	36,800
		Grand total . .	$2,547,000	203,760

(*) mva = millions of va

Equipment Feeding Substations. The equipment feeding substations involves the equipment and accessories given below:

Equipment Feeding Substations

Units	Description	Cost
6	200-kva portable substations one spare unit	$ 48,000
4	900-kva portable substations one spare unit	60,000
General:		
	Trailing Cable and Accessories; three 2,000-ft pieces, Trifasic No. 6 AWG, shielded, Butyl synthetic rubber (15,000 v). Current capacity, 76 amp. Estimated cost, $6.00/100 amp-ft (including cable couplers and instalation). Total cost= 2,000 ft x x 3 cables x 76 amp x $6.0/100 amp-ft.	30,400
	Trailing Cable and Accessories; five 2,000-ft pieces, Trifasic No. 8 AWG, shielded, butyl synthetic rubber (15,000 v). Current capacity, 46 amp. Estimated cost, $4.00/100 amp-ft (including cable couplers and instalation). Total cost=2,000 ft x x 3 cables x 46 amp x $4.0/100 amp-ft.	18,400
	Pole line feeder around pit. two 4,300-ft pieces. Cable trifasic No. 500 MCM, 300 amp, shielded (15,000 v). Installed cost $5.00/100 amp-ft. Total cost=8,600 ft x 5 cables x x 300 amp x $5.00/100 amp-ft.	130,000
Others	(miscellaneous)	5,000
	Total	$291,800

Summary of Costs

Linking Substation	$ 94,080
Transmission Line	1,901,800
Mine Substation	203,760
Feeding Substations	291,800
Grand total electric facilities	$2,199,640

Mining Preparation

The preparation of the pit includes the construction of the roads as proposed on the surface map. These roads will permit the access of the equipment to the stripping area, and later to the production benches. The total length of the roads amounts approximately to 4.36 miles (7 km), and the estimated cost per mile of road amounts to $ 9,650.00.

Total estimated cost of roads = 4.36 miles x x 9,650.0/mile = $42,000.00

The time involved in preparation is estimated as half a year; during this period, the electrical facilities should be installed for the starting of the stripping operations.

Stripping

The material to be stripped corresponds to benches OO, through BB. These areas are represented in Fig. 8, to 21. The total stripping (134 million tons) is projected to be done in a period of approximately 1,036 days. During this period the metallurgical plant is estimated to be finished, in order to start with the normal production.

Calculation of equipment for stripping is given in Appendix C, and the results are summarized below.

Units	Description
5	Drilling rigs capable of performing 7 3/8-in. holes.
3	16-cu-yd electric power shovels
2	8-cu-yd loaders
1	5-cu-yd loader
19	50-ton trucks, including two spare units.

Production

The production of ore takes place from benches AA to E, at a continuous rate of 30,000 tons per day. In order to accomplish this plan, it is necessary to strip an adequate amount of overburden to expose ample ore for daily production needs.

Table 6, shows the waste/ore ratio for the different benches. It has been prepared to auxiliate in calculation of production equipment. For the production section of the pit, the overall waste/ore ratio becomes 1.54/1.00.

Table 6. Waste/Ore ratio

Bench	Ore Area sq ft	Waste Area sq ft	Waste/Ore Ratio	Cumulative Waste/Ore Ratio
AA	430,000	3,469,024	8:1	8:1
Z	559,520	3,469,024	6.2:1	7:1
Y	1,431,080	2,549,820	1.8:1	4:1
X	1,280,440	2,895,516	2.3:1	3.4:1
W	1,512,160	2,829,880	1.9;1	3.0:1
V	1,732,360	2,808,360	1.6:1	2.6:1
U	1,557,800	2,455,432	1.5:1	2.4:1
T	2,281,120	1,813,060	0.8:1	2.1:1
S	2,259,600	1,746,348	0.77:1	1.85:1
R	1,506,400	2,285,424	1.51:1	1.81:1
Q	1,614,000	1,914,204	1.18:1	1.75:1
P	1,237,400	2,077,756	1.69:1	1.75:1
O	974,856	1,975,536	2.03:1	1.76:1
N	309,888	2,268,208	7.34:1	1.85:1
M	1,108,200	1,076,000	0.97:1	1.81:1
L	1,608,620	269,000	0.17:1	1.68:1
K	1,076,000	538,000	0.50:1	1.62:1
J	1,275,060	302,356	0.23:1	1.55:1
I	267,924	459,452	1.71:1	1.55:1
H	275,456	275,456	1.00:1	1.53:1
G	234,568	194,756	0.83:1	1.53:1
F	0	335,712	-	1.55:1
E	139,380	111,904	0.80:1	1.54:1

Planning Production

From Table 6, it is seen that the overall waste/ore ratio is 1.54:1. This means that stripping equipment with a capacity 1.54 times the capacity of the production equipment is sufficient for this work. Production is estimated to take place at a rate

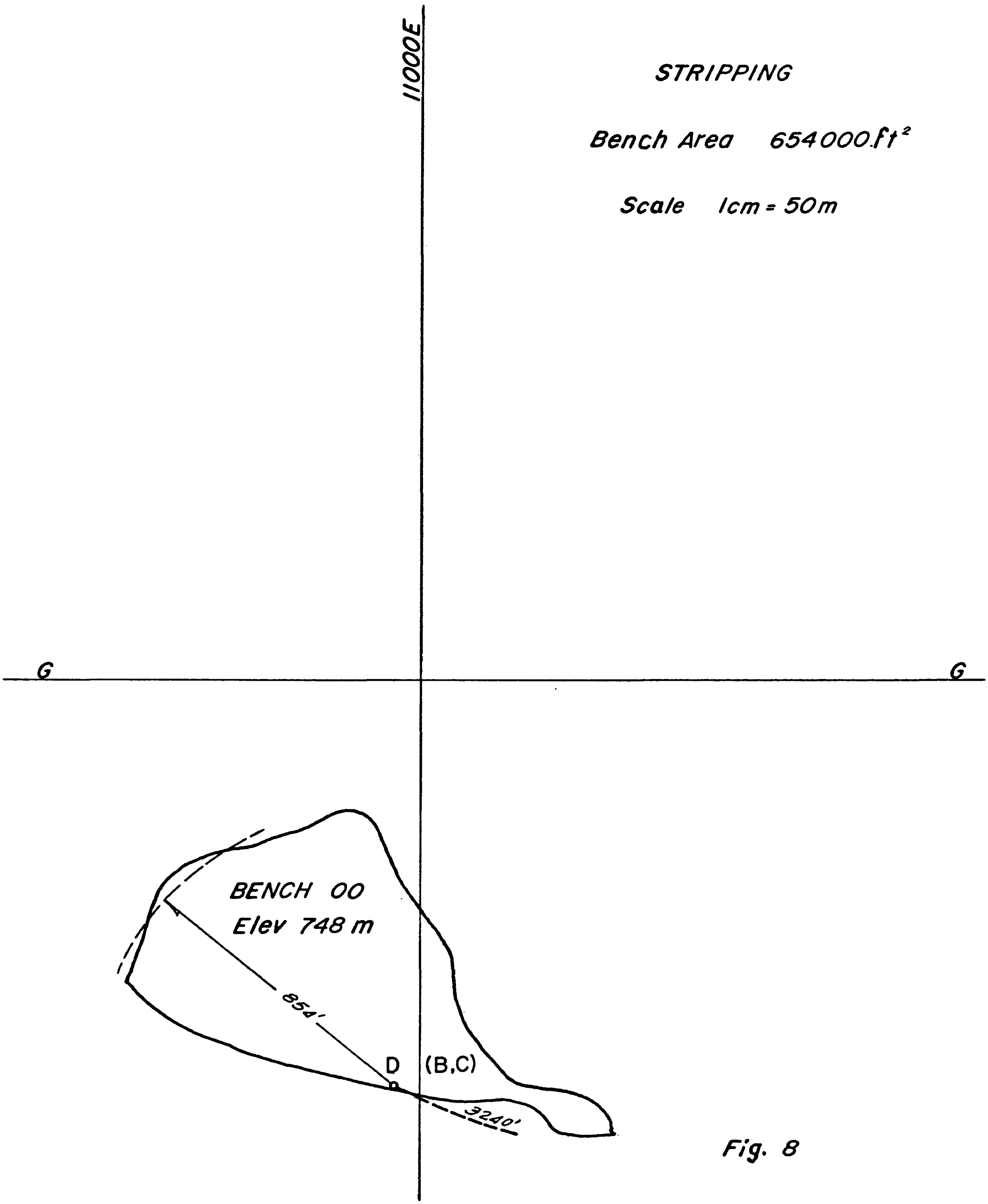

11000E
STRIPPING
Bench Area 654000.ft²
Scale 1cm = 50m
G
G
BENCH 00
Elev 748 m
854'
D
(B,C)
3240'

Fig. 8

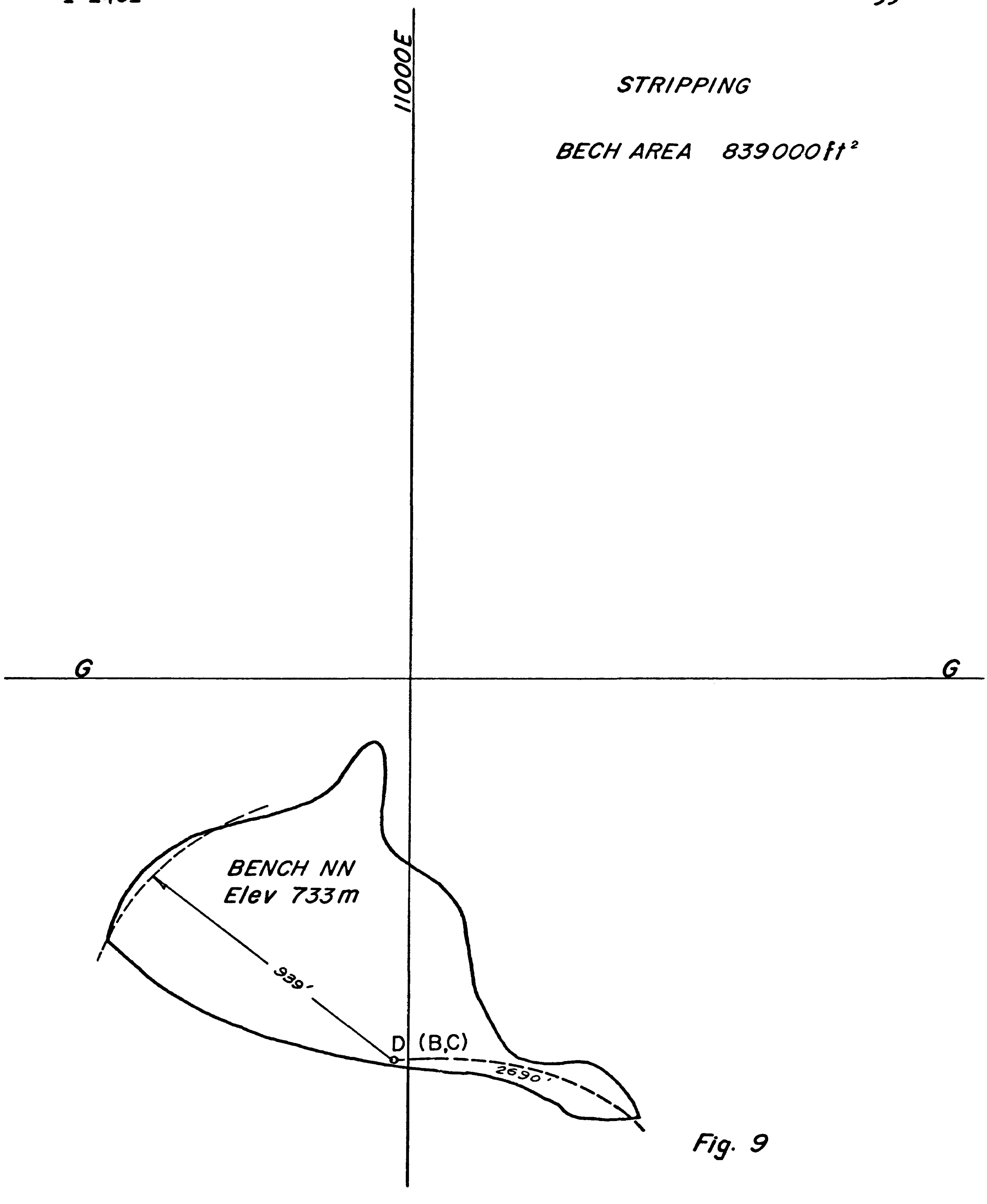

11000E
STRIPPING
BECH AREA 839000 ft²
G
G
BENCH NN
Elev 733m
939'
D (B,C)
2690'

Fig. 9

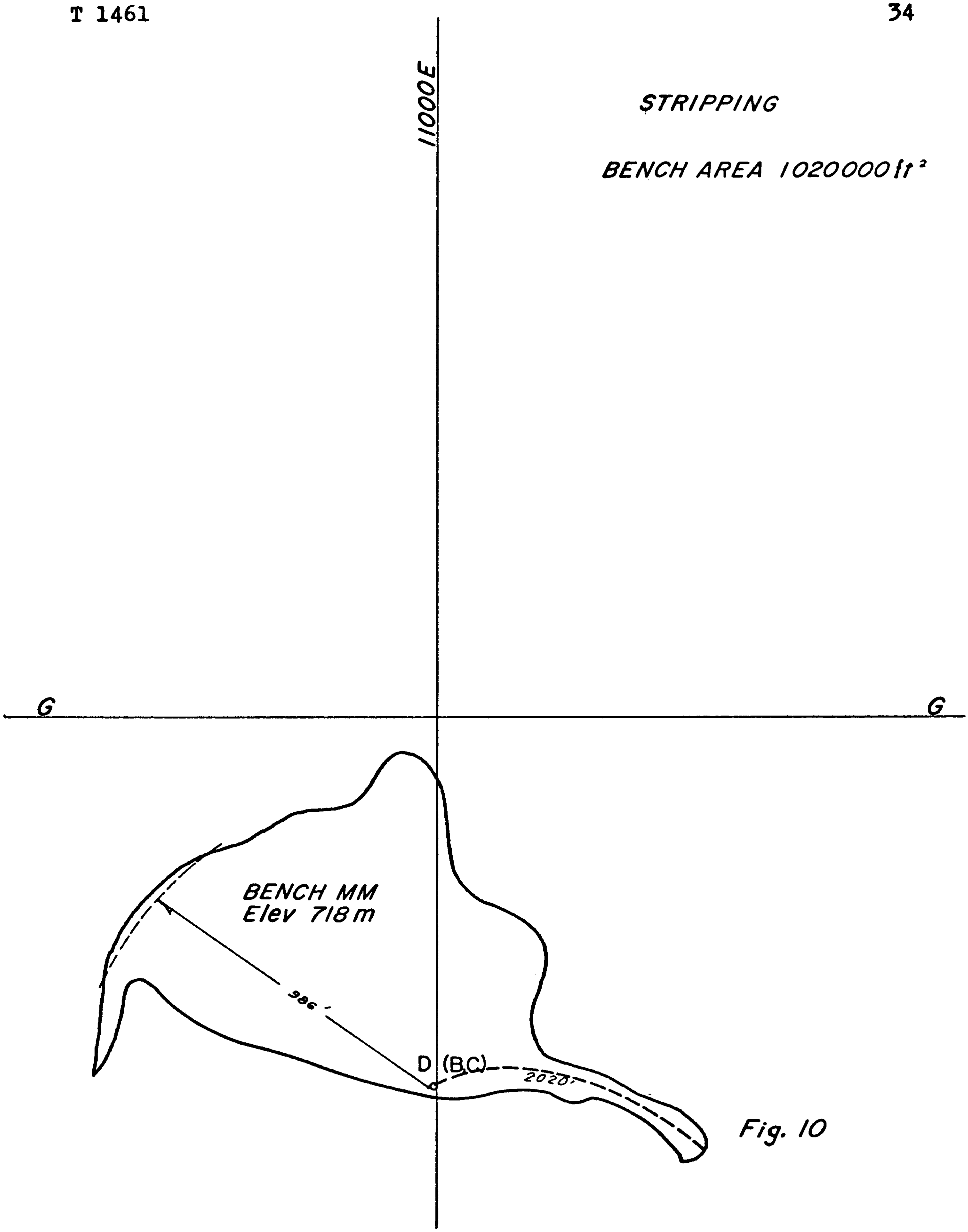

11000E
STRIPPING
BENCH AREA 1020000 ft²
G
G
BENCH MM
Elev 718 m
986'
D (B.C)
2020'

Fig. 10

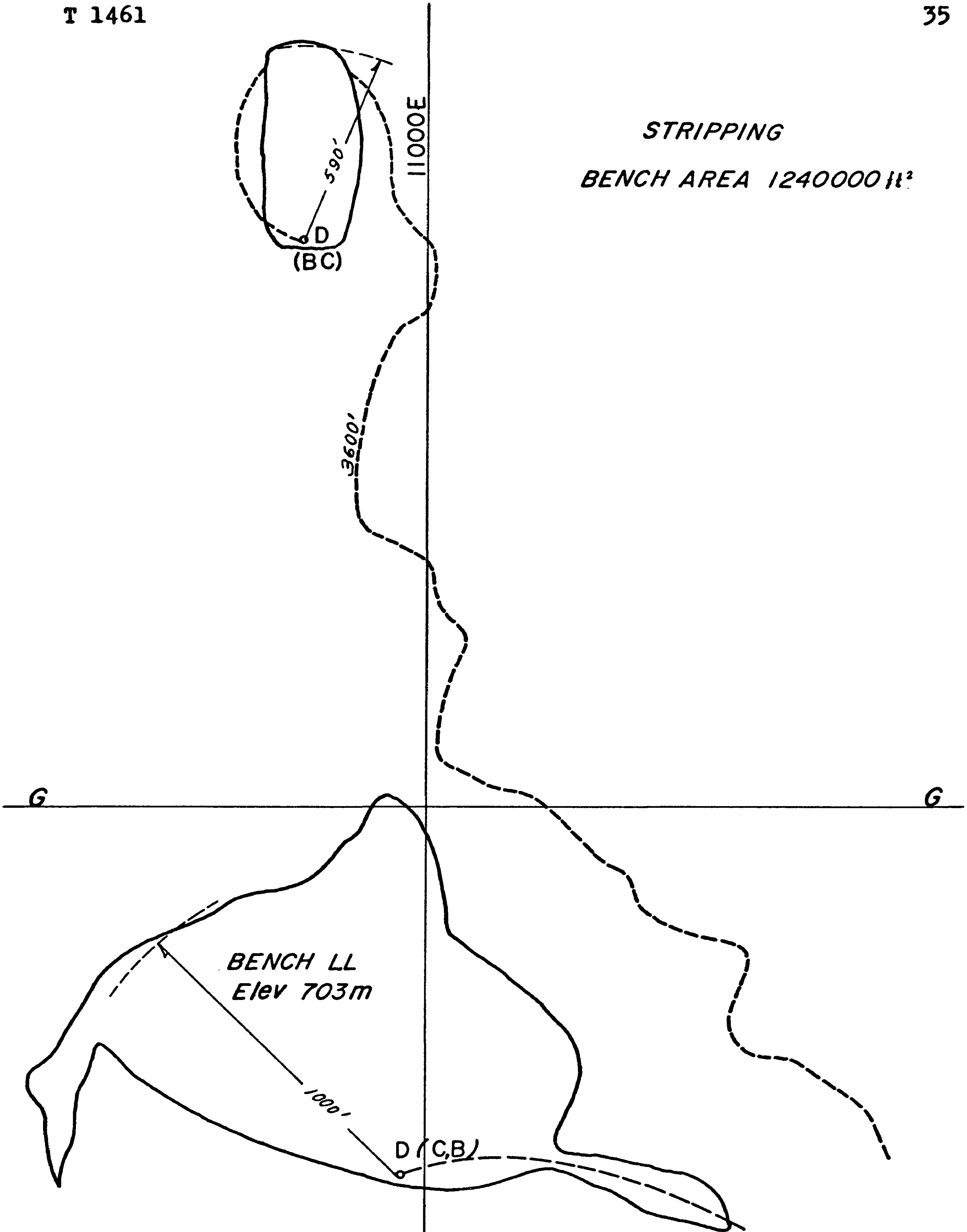

STRIPPING
BENCH AREA 1240000 ft²
11000E
590'
D
(BC)
3600'
G
G
BENCH LL
Elev 703m
1000'
D (C,B)

Fig. 11

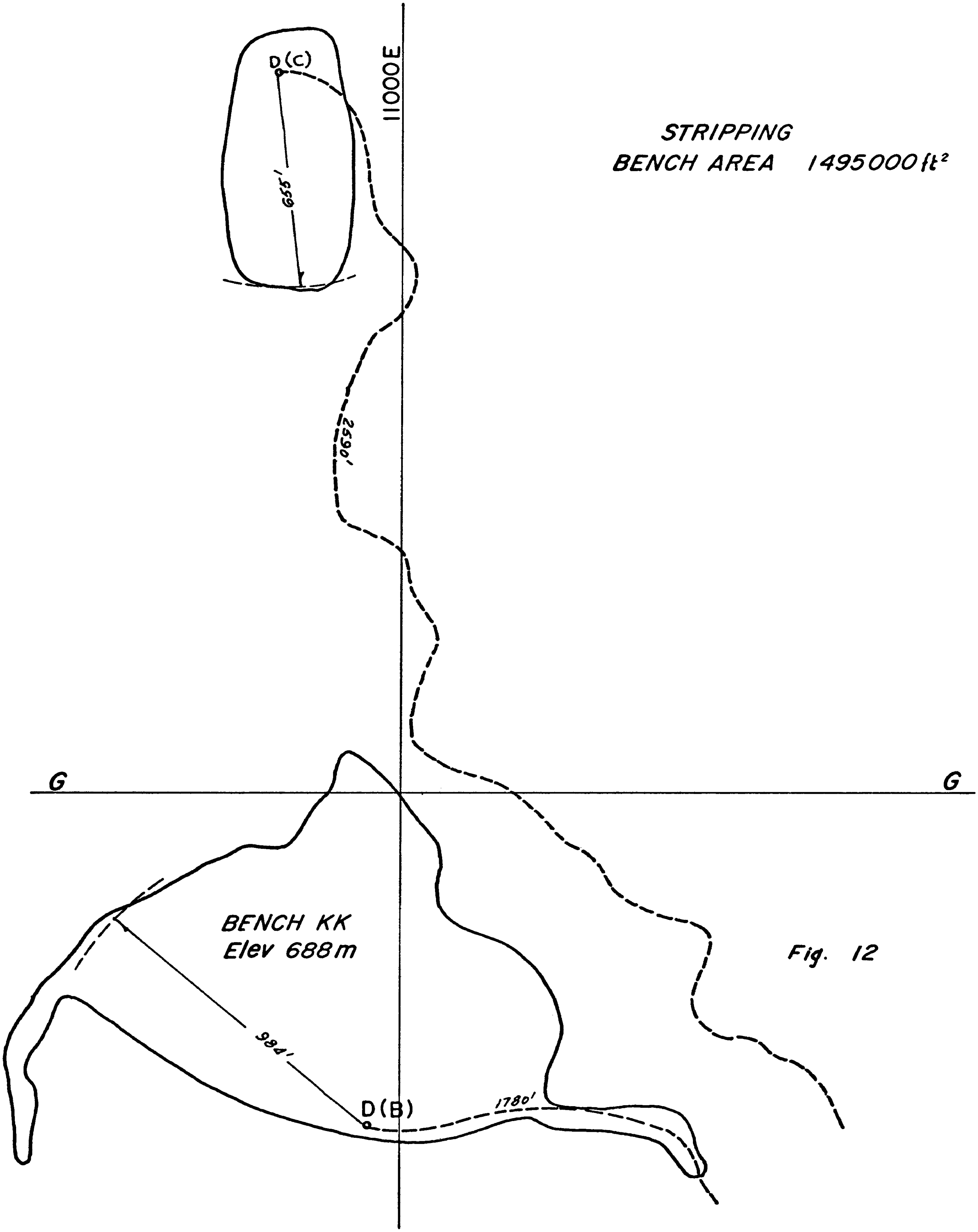

11000E
D(C)
655'
STRIPPING
BENCH AREA 1495000 ft²
2590'
G
G
BENCH KK
Elev 688 m
984'
D(B)
1780'

Fig. 12

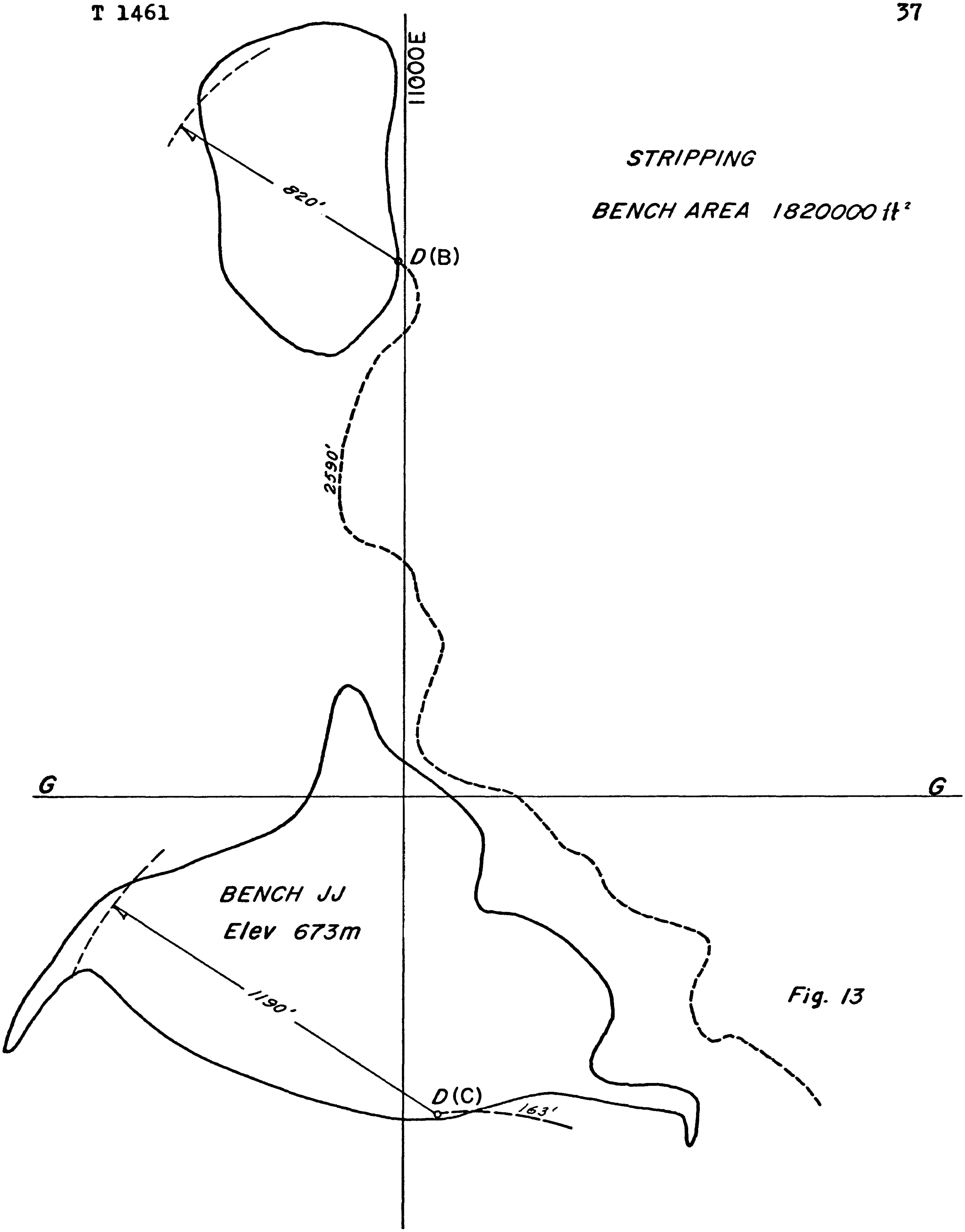

11000E
STRIPPING
BENCH AREA 1820000 ft²
820'
D(B)
2590'
G
G
BENCH JJ
Elev 673m
1190'
D(C)
163'

Fig. 13

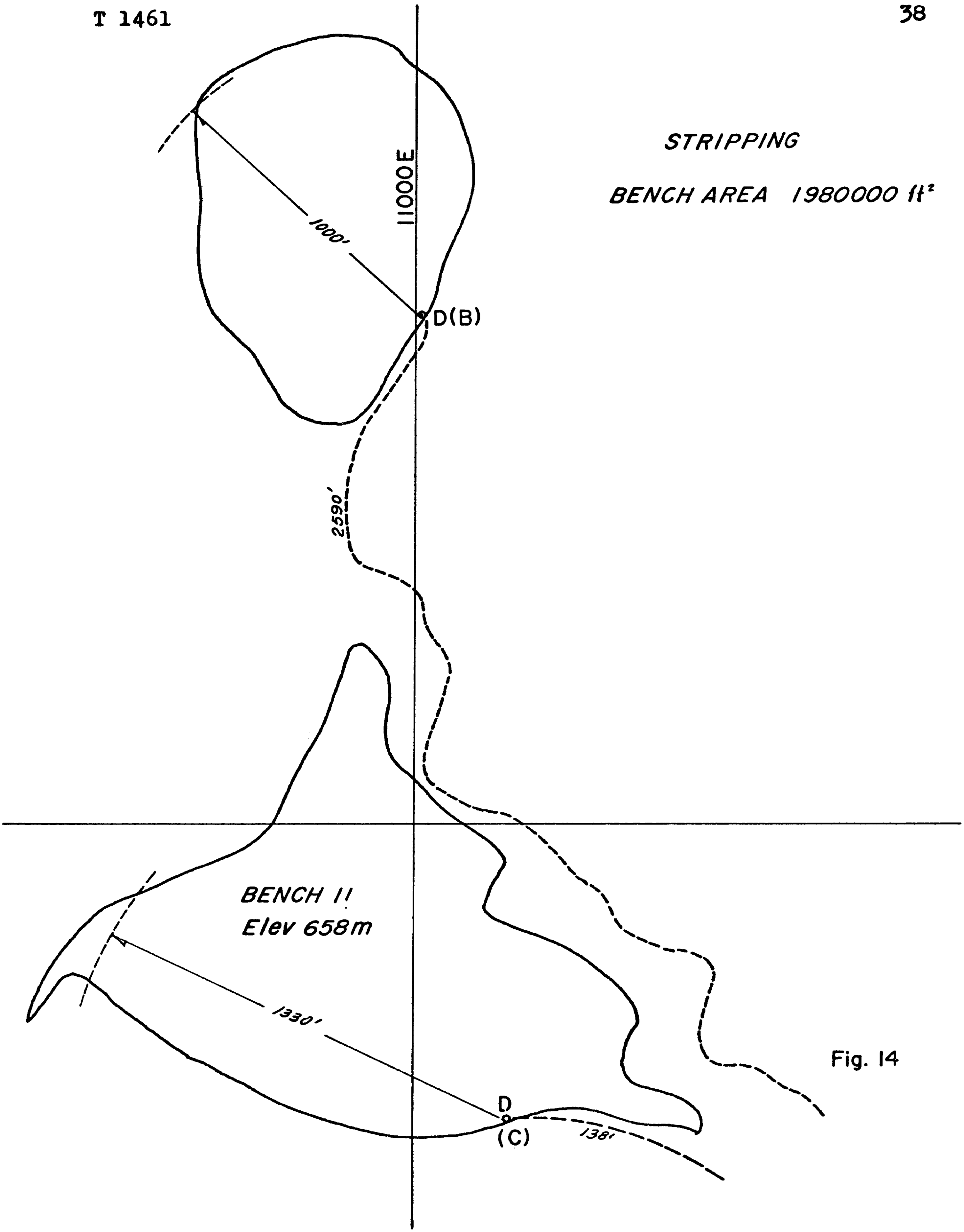

STRIPPING
BENCH AREA 1980000 ft²
11000E
1000'
D(B)
2590'
BENCH 11
Elev 658m
1330'
D
(C)
138'

Fig. 14

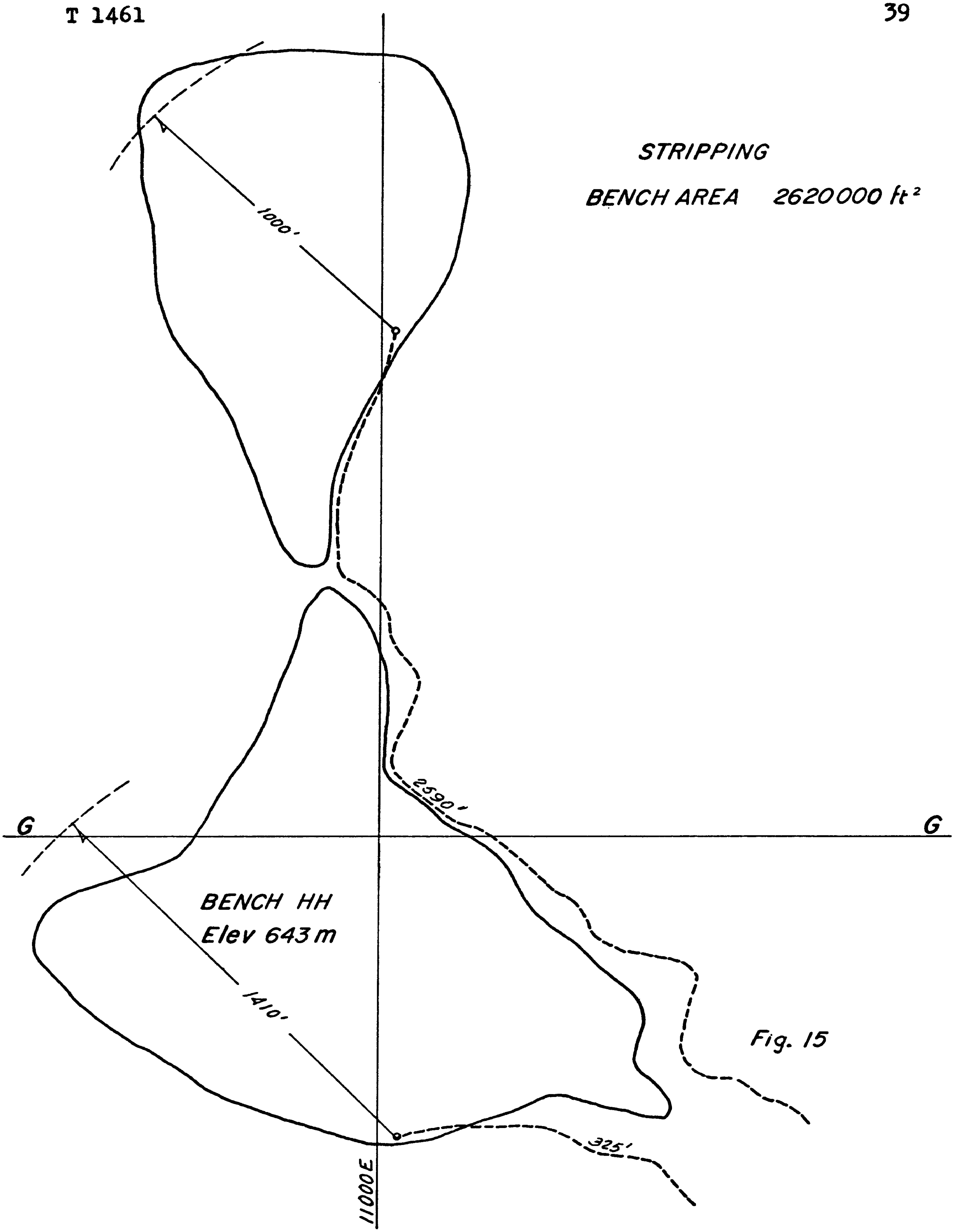

STRIPPING
BENCH AREA 2620000 ft²
1000'
2590'
G
G
BENCH HH
Elev 643 m
1410'
325'
11000E

Fig. 15

STRIPPING
BENCH AREA 2852080 ft²

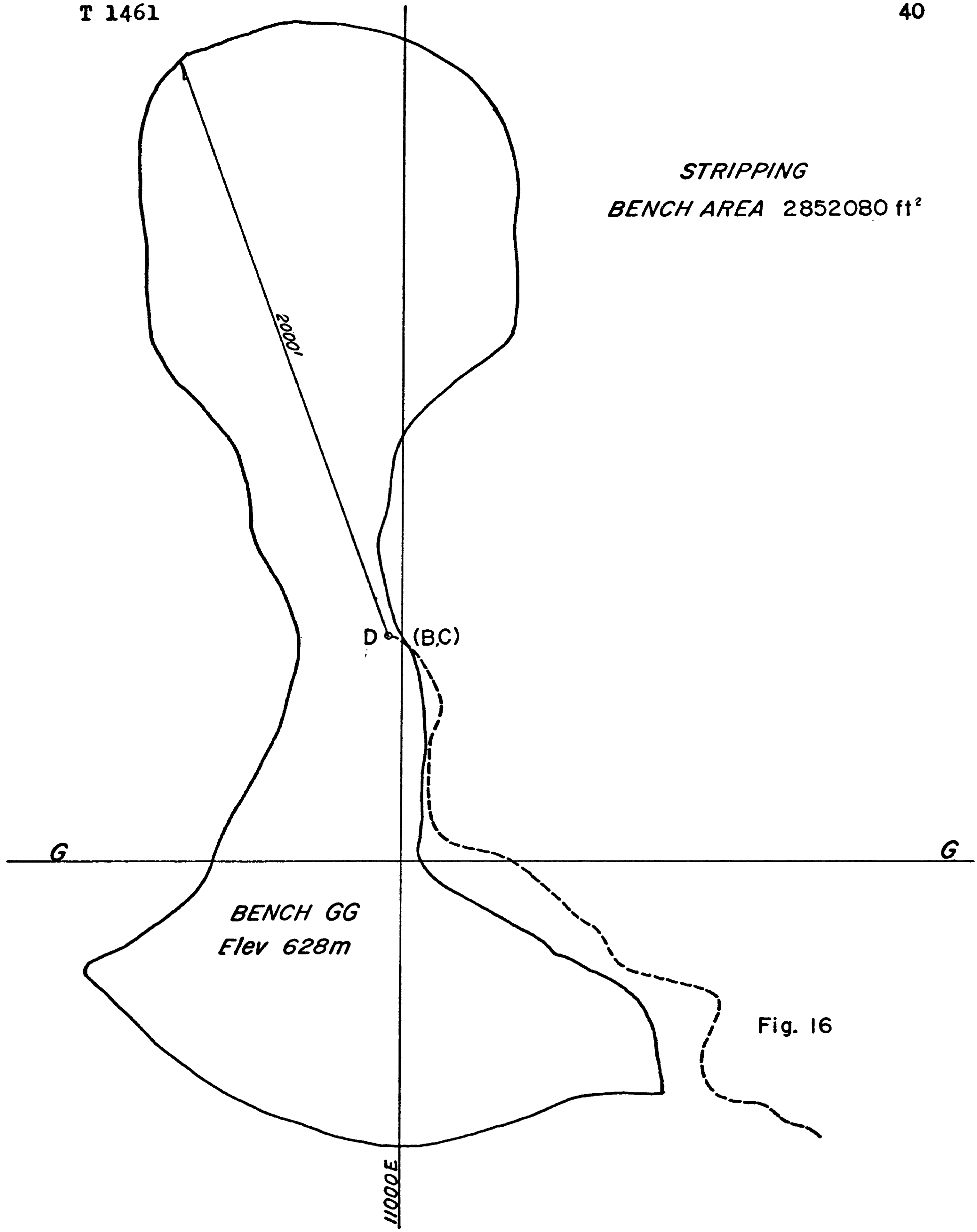
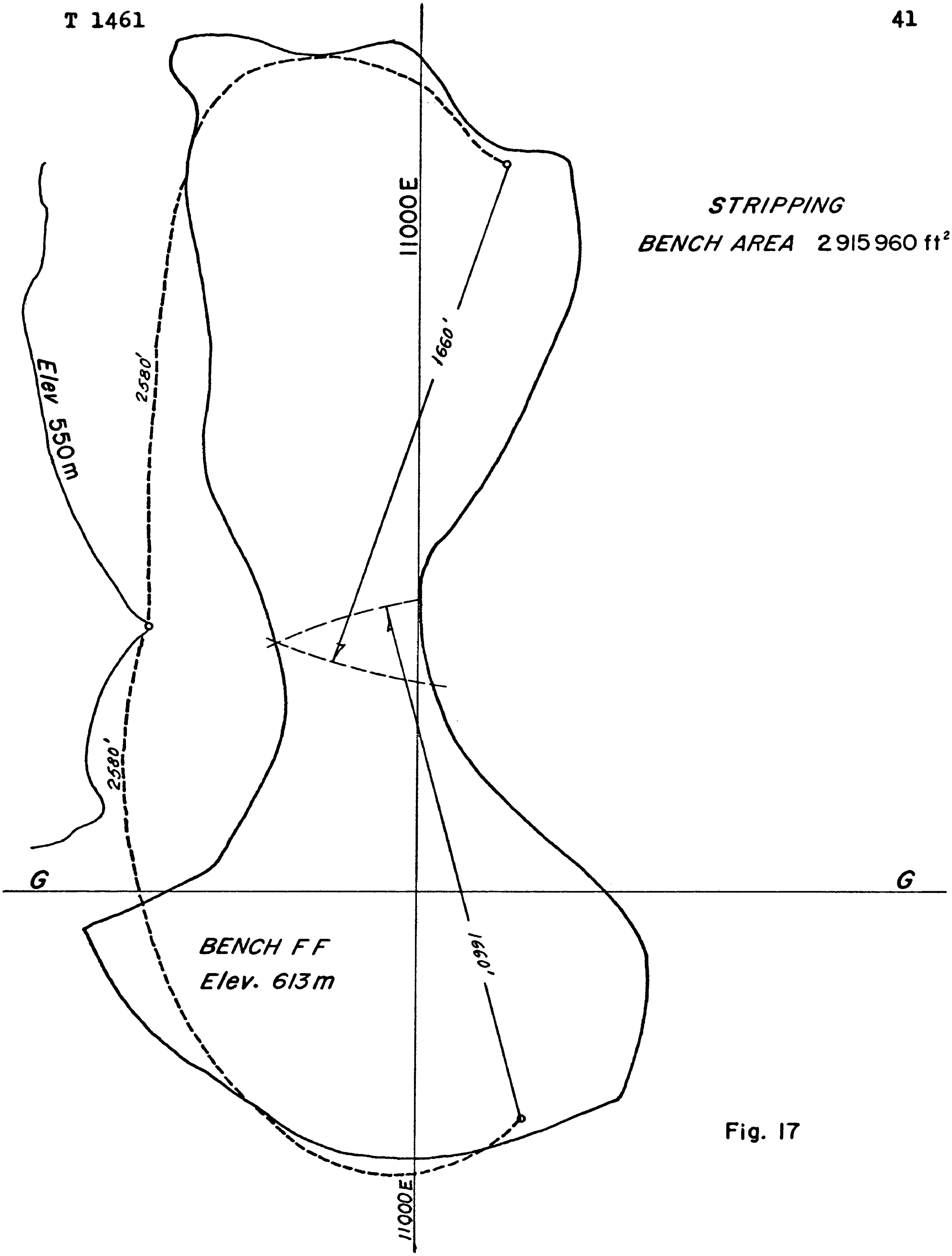


Fig. 16



STRIPPING
BENCH AREA 2 915 960 ft²
11000E
1660'
2580'
Elev 550m
G
G
BENCH FF
Elev. 613m
11000E

Fig. 17

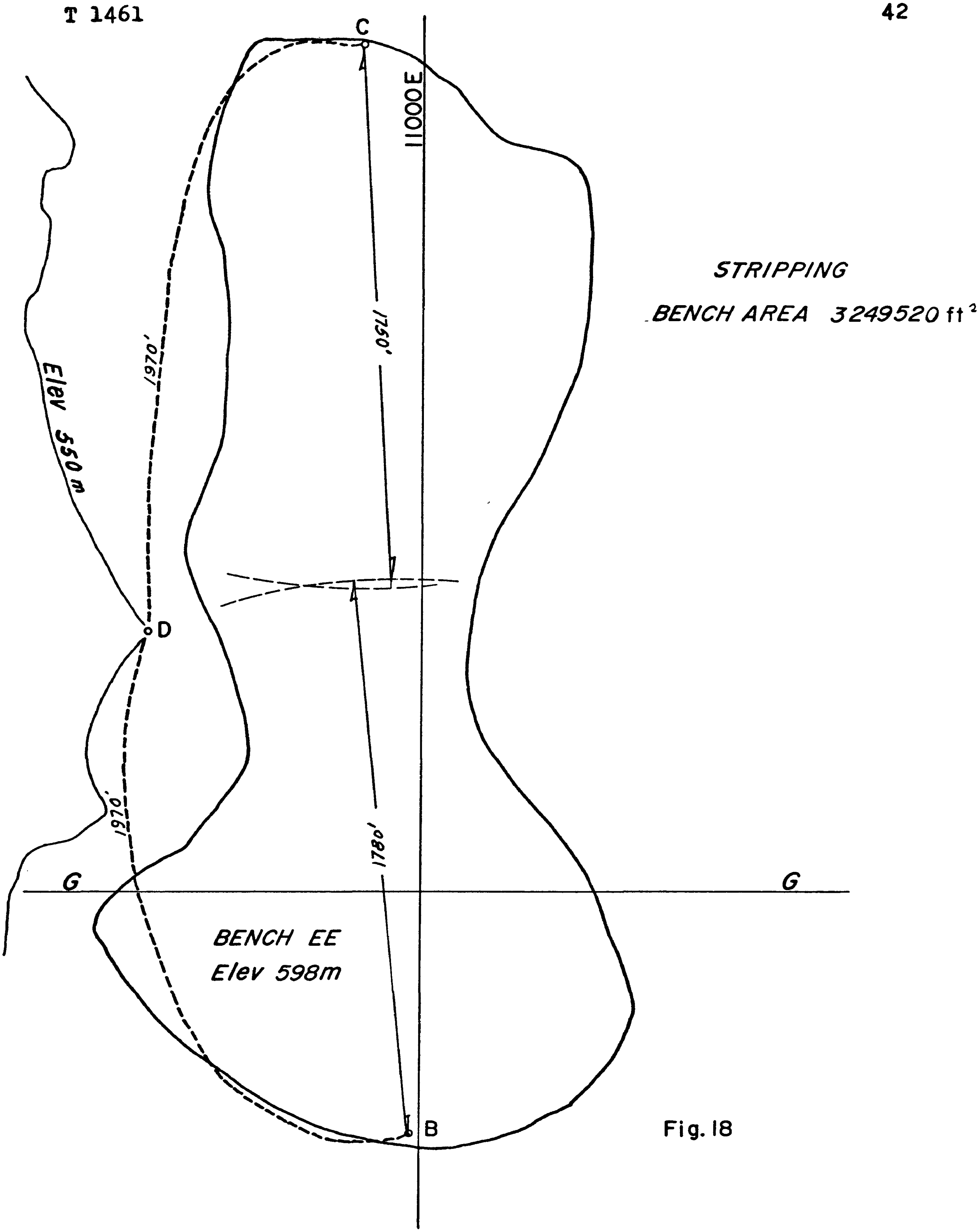

C
11000E
STRIPPING
BENCH AREA 3249520 ft²
1750'
1970'
Elev 550 m
D
1970'
1780'
G
G
BENCH EE
Elev 598m
B

Fig. 18

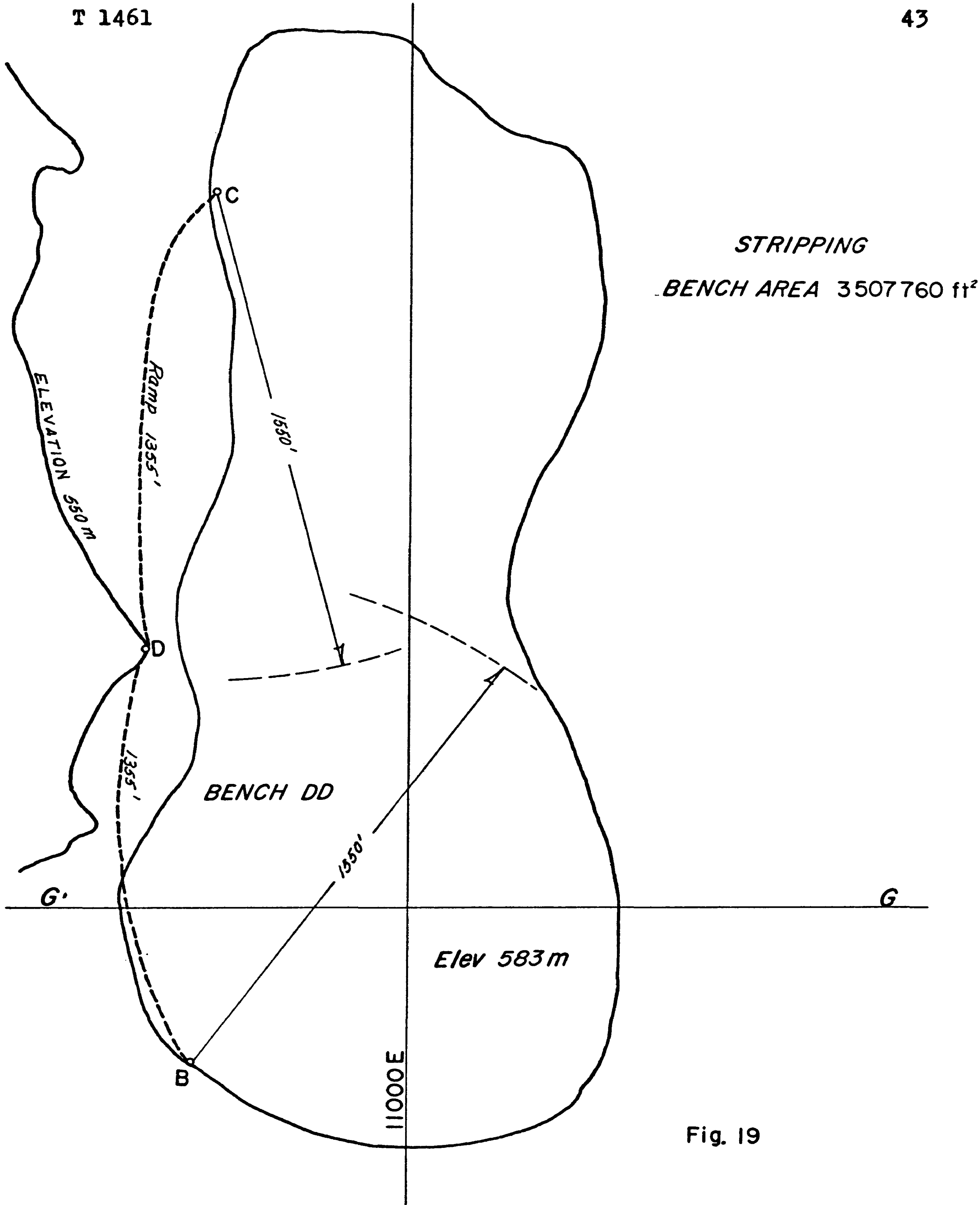

STRIPPING
BENCH AREA 3507760 ft²
C
Ramp 1355'
ELEVATION 550 m
1550'
D
1355'
BENCH DD
1350'
G'
G
Elev 583 m
B
11000 E

Fig. 19

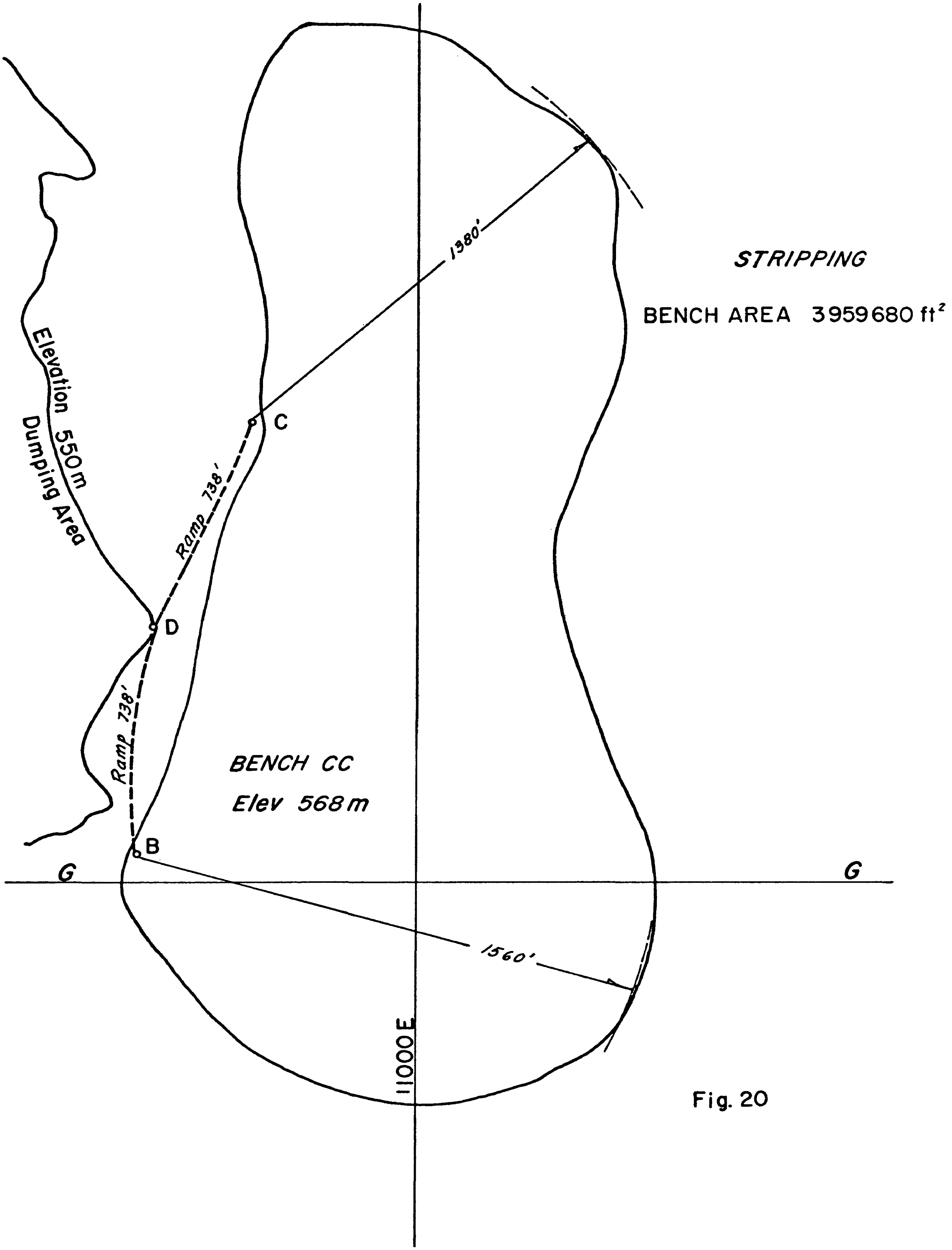

STRIPPING
BENCH AREA 3959680 ft²
1380'
Elevation 550m
Dumping Area
C
Ramp 738'
D
Ramp 738'
BENCH CC
Elev 568m
B
G
G
1560'
11000E

Fig. 20

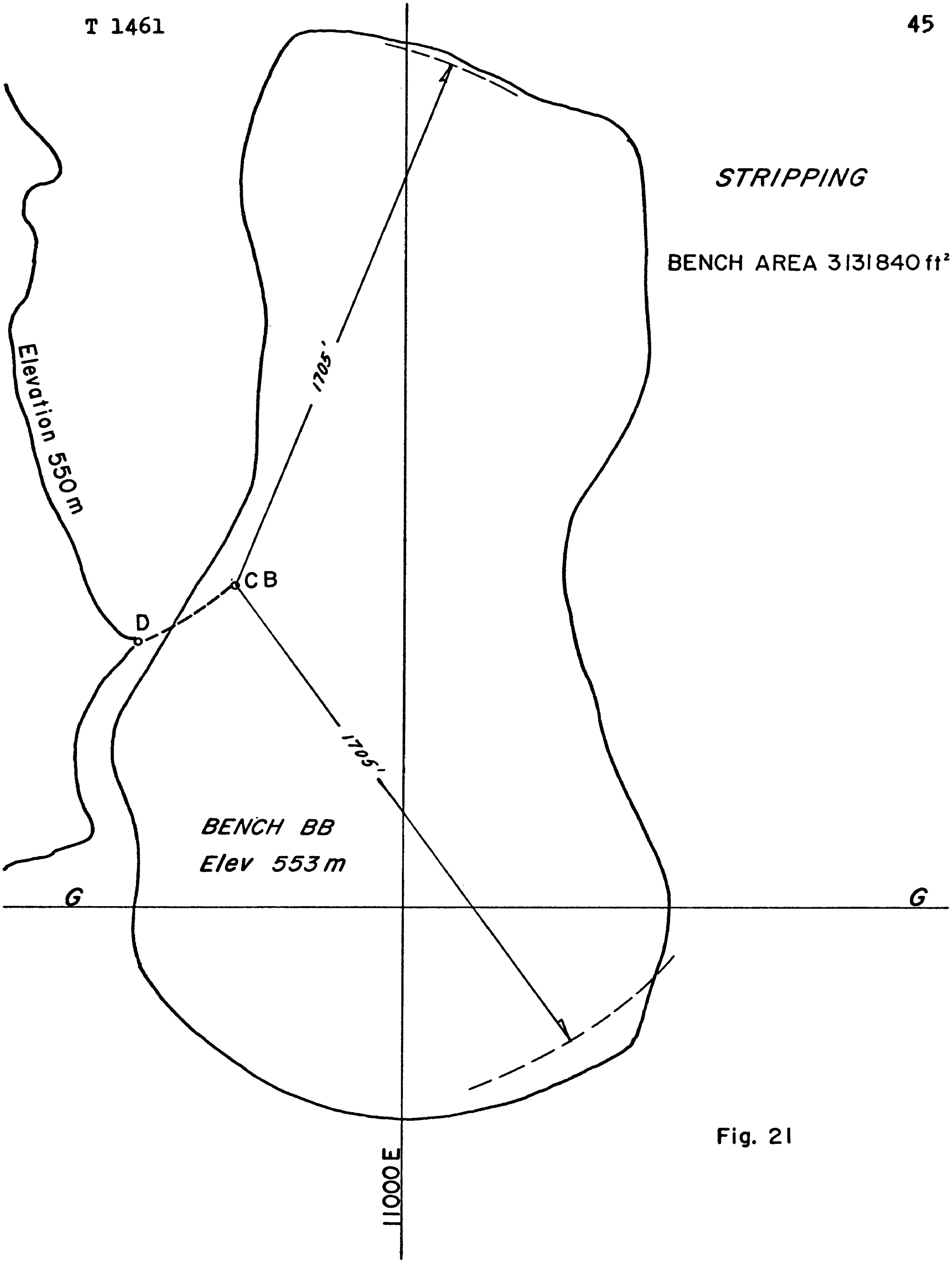

STRIPPING
BENCH AREA 3131840 ft²
Elevation 550 m
1705'
C B
D
1705'
BENCH BB
Elev 553 m
G
G
11000 E

Fig. 21

of 30,000 tons per day. The capacity of each 16-cu-yd shovel as calculated in Appendix C, is 33,000 tons per day. This fact indicates that one single shovel can produce the daily ore requirements. However, production can not be so efficient, because of the need of ore blending. In this respect, two alternatives exist; one is the selling of one big shovel and replacing it by two smaller ones. The other, is the acceptance of a less efficient operation with the big shovel complemented with one front-end loader. For this set-up, the big shovel can operate in production.

In relation to stripping, two 16-cu-yd shovels are able to give 2.2 times the production rate ($2.2 > 1.54$), and under these conditions, two 16-cu-yd shovels are more than sufficient for the required work. Smaller shovels are not recommended because it is preferable to have sufficient stripping capacity in order to sustain the daily production under critical conditions.

Table 7, in combination with Fig. 23 to 34, shows the production program for the 12 years of life of the mine. For the first year the sequence of exploitation is as follows:

a). The mineralized area AA(1) (430,000 sq ft), is extracted at the same time that an amount of 948,000 ft^2 of waste is stripped.

b). The ore in bench AA is now exhausted, so production of ore comes from bench Z (559,520 sq ft), at the same time that 509,000 sq ft of waste are stripped on bench AA.

c). When the ore in bench Z is exhausted, production of ore continues from bench Y (1,171,080 sq ft), at the time that 1,111,308 sq ft of waste are stripped on bench AA, 1,090,000 sq ft on bench Z, and 382,000 on bench Y.

Numbers enclosed in brackets, indicate sequence of operations.

That is, operation (1) precedes operation (2), and so forth.

Common numbers on different benches indicate simultaneous production-stripping operations on the corresponding benches, and the changing shape of the pit through annual periods can be followed in the corresponding figures.

The production-hauling equipment calculation is developed in Appendix C, and the summary of this equipment is shown in Table 11 of the same Appendix. From this table, the number of hauling units can be chosen as 19, and under these conditions no change in hauling equipment is necessary when passing from stripping to production operations.

Trucks utilized in transportation of concentrates are estimated independently in the market and transportation section.

Labor

Labor can be recruited in the area, but a special training program must be developed to enable the workers to carry out the current operations. The number of people for mining operations not including concentrate-truck drivers (Appendix D) is estimated as 382 men distributed as follows:

Staff and labor force	Men
Truck and pick-up drivers	80
Drillers	45
Shovel operators and helpers	27
Loader operators	9
Grader operators	9
Maintenance estimated as two men per operating unit	108
Blasters and other	32
Total operating labor	310
Geology and surveying	12
Mine planning	30
Laboratory	30
Total	382

Table 7. Production Program

Year	Bench	Ore Area sq ft	Waste Area sq ft	Ore Tons	Waste Tons	Cumulative Waste Tons	Cumulative Ore Tons
1	AA(1)	430,000	948,000	1,794,850	3,945,000	3,945,000	1,794,850
	AA(2)	-	509,000	-	2,120,000	6,065,000	-
	AA(3)		1,111,308	-	4,620,000	10,685,000	-
	Z(2)	559,520	720,000	2,332,915	2,998,000	13,683,000	4,125,765
	Z(3)	-	1,090,000	-	4,540,000	18,223,000	-
	Y(3)	1,171,080	382,000	4,874,235	1,577,000	19,800,000	9,000,000
2	AA(4)	-	629,908	-	2,620,000	22,420,000	-
	AA(5)	-	270,808	-	1,130,000	23,550,000	-
	Z(4)	-	679,674	-	2,825,000	26,375,000	-
	Z(5)	-	499,675	-	2,080,000	28,455,000	-
	Y(3)	260,000	338,000	1,092,644	1,423,000	29,878,000	10,092,644
	Y(4)	-	328,000	-	1,365,000	31,243,000	-
	Y(5)	-	909,820	-	3,785,000	35,028,000	-
	X(4)	1,280,440	1,000,000	5,338,786	4,160,000	39,188,000	15,431,432
	X(5)	-	103,807	-	431,000	39,619,000	-
	W(5)	618,960	-	2,568,568	-	39,619,000	18,000,000
3	Z(6)	-	479,675	-	1,994,000	41,613,000	-
	Y(6)	-	592,000	-	2,460,000	44,073,000	-
	X(5)	-	570,193	-	2,374,000	46,447,000	-
	X(6)	-	769,516	-	3,200,000	49,647,000	-
	W(5)	898,200	369,000	3,757,221	1,537,000	51,184,000	21,757,221
	W(6)	-	1,010,880	-	4,200,000	55,384,000	-
	V(6)	1,261,800	775,000	5,242,779	3,230,000	58,614,000	27,000,000

Table 7. (cont'd.)

Year	Bench	Ore Area sq ft	Waste Area sq ft	Ore Tons	Waste Tons	Cumulative Waste Tons	Cumulative Ore Tons
4	X(7)	-	452,000	-	1,880,000	60,494,000	-
	W(7)	-	970,000	-	4,045,000	64,539,000	-
	V(7)	470,560	1,168,000	1,980,285	4,860,000	69,399,000	28,980,285
	U(7)	1,167,800	934,000	6,953,881	3,884,000	73,283,000	35,934,166
5	V(8)	-	865,000	-	3,600,000	76,883,000	-
	U(8)	-	1,521,432	-	6,340,000	83,223,000	-
	T(8)	2,281,120	1,813,060	9,511,115	7,550,000	90,773,000	45,445,281
6	S(9)	2,259,600	1,746,348	9,421,387	7,260,000	98,033,000	54,866,668
7	R(10)	1,506,400	2,285,424	6,280,925	9,530,000	107,563,000	61,147,593
	Q(11)	454,520	1,000,000	1,852,407	4,160,000	111,723,000	63,000,000
8	Q(11)	1,159,480	914,204	4,877,155	3,800,000	115,523,000	67,877,155
	P(12)	1,237,400	2,077,756	5,159,331	8,650,000	124,173,000	73,036,486
9	O(13)	974,856	1,975,536	4,064,656	8,230,000	132,403,000	77,101,142
	N(14)	309,888	2,118,208	1,292,076	8,820,000	141,223,000	78,393,218
	M(15)	390,000	-	1,622,000	-	-	80,015,218
10	N(14)	-	350,000	-	1,460,000	142,683,000	-
	M(15)	718,280	860,000	2,990,000	3,580,000	146,263,000	83,005,218
	L(16)	1,608,620	269,000	6,707,130	1,120,000	147,383,000	89,712,348
11	K(17)	1,076,000	538,000	4,486,375	2,240,000	149,623,000	94,198,723
	J(18)	1,275,000	302,356	5,316,354	1,260,000	150,883,000	99,915,077
12	I(19)	267,924	459,452	1,117,107	1,910,000	152,793,000	100,632,184
	H(20)	275,456	275,456	1,148,512	1,148,512	153,941,000	101,780,696
	G(21)	234,568	194,756	978,030	810,000	154,751,000	102,758,726
	F(22)	-	335,712	-	1,396,000	156,147,000	-
	E(23)	251,784	111,904	1,060,476	1,356,000	157,503,000	103,819,202

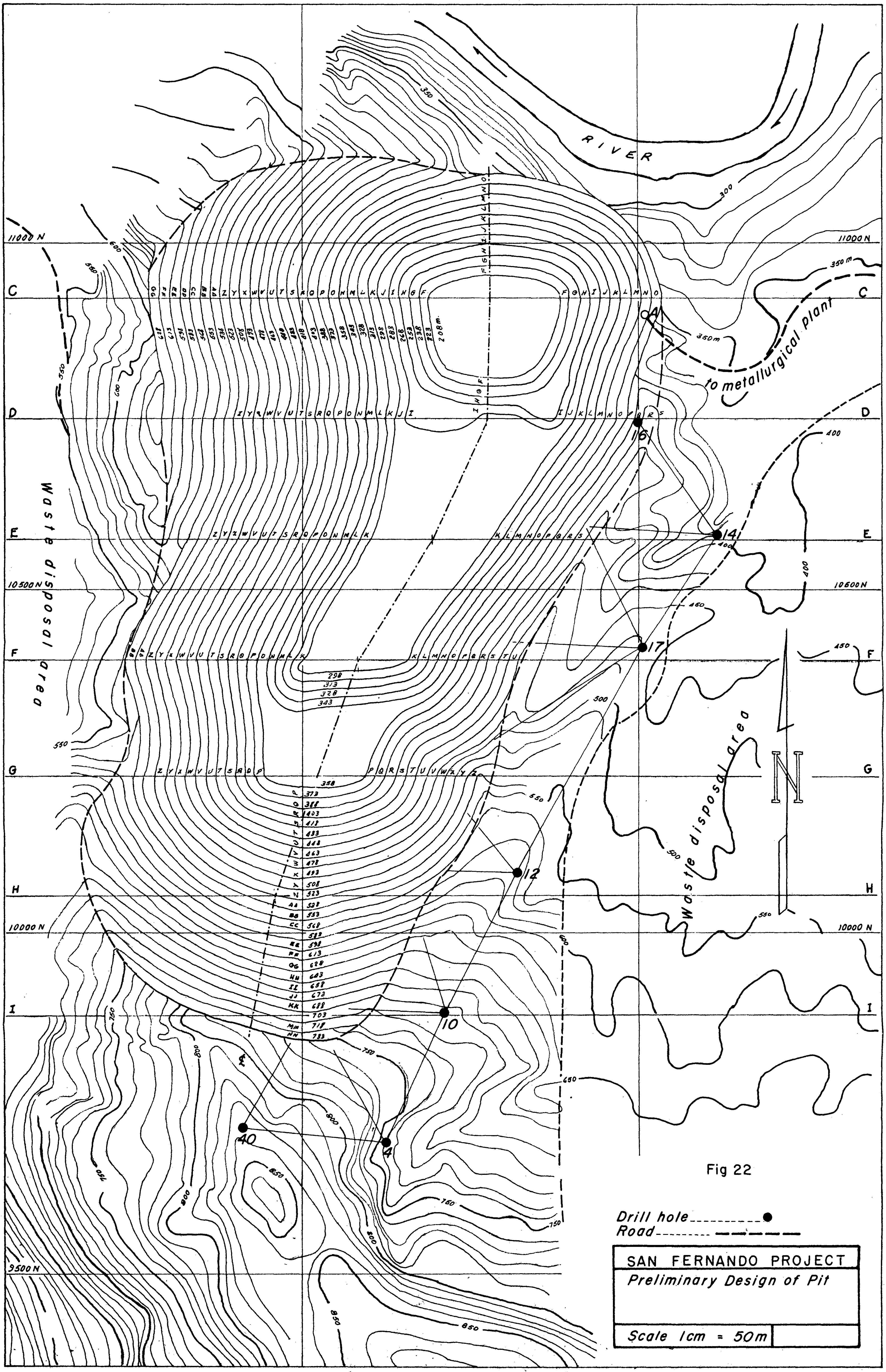

RIVER
to metallurgical plant
Waste disposal area
Waste disposal area
N
11000 N
10500 N
10000 N
9500 N
C
D
E
F
G
H
I
Drill hole
Road
SAN FERNANDO PROJECT
Preliminary Design of Pit
Scale 1cm = 50m

Fig 22

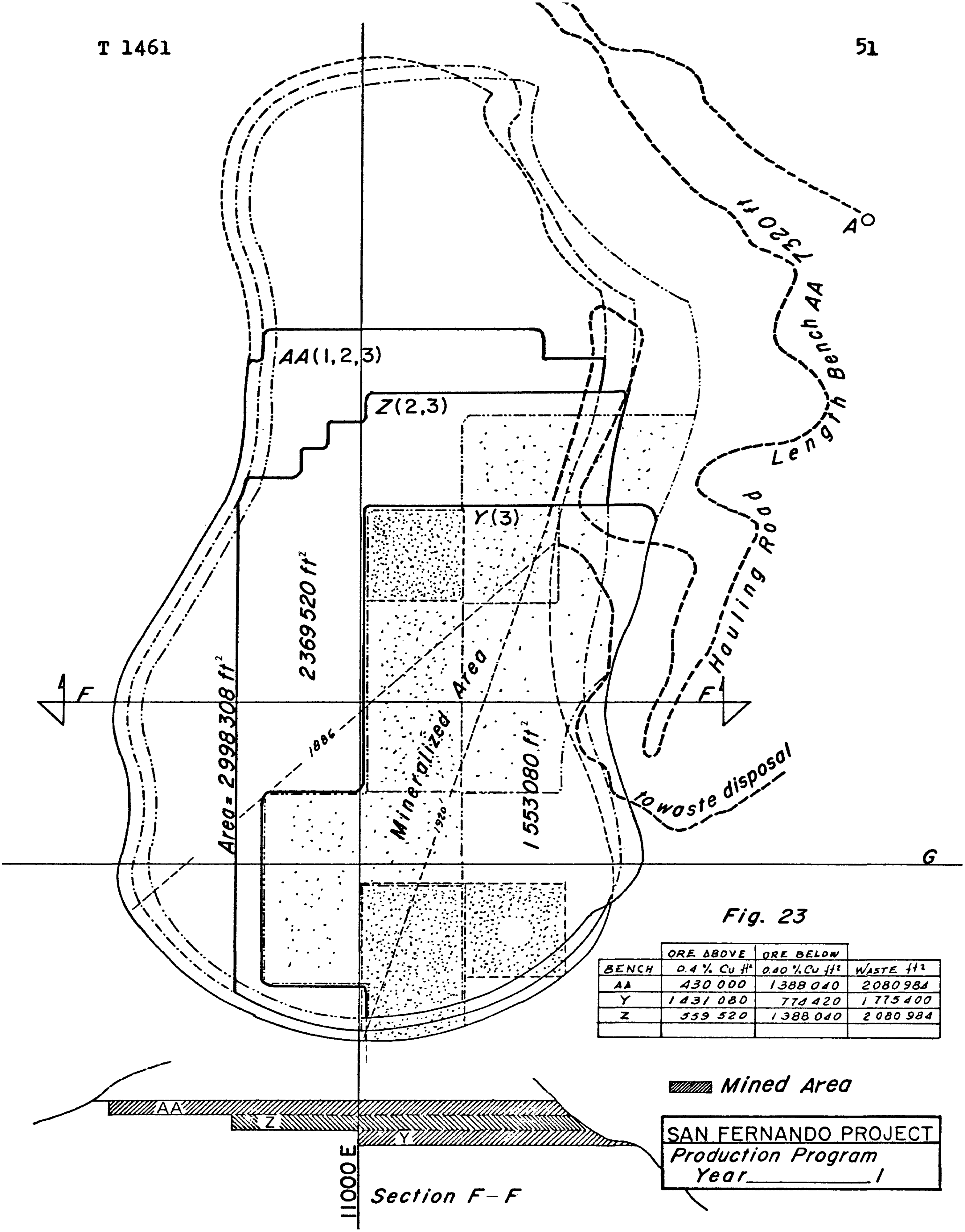
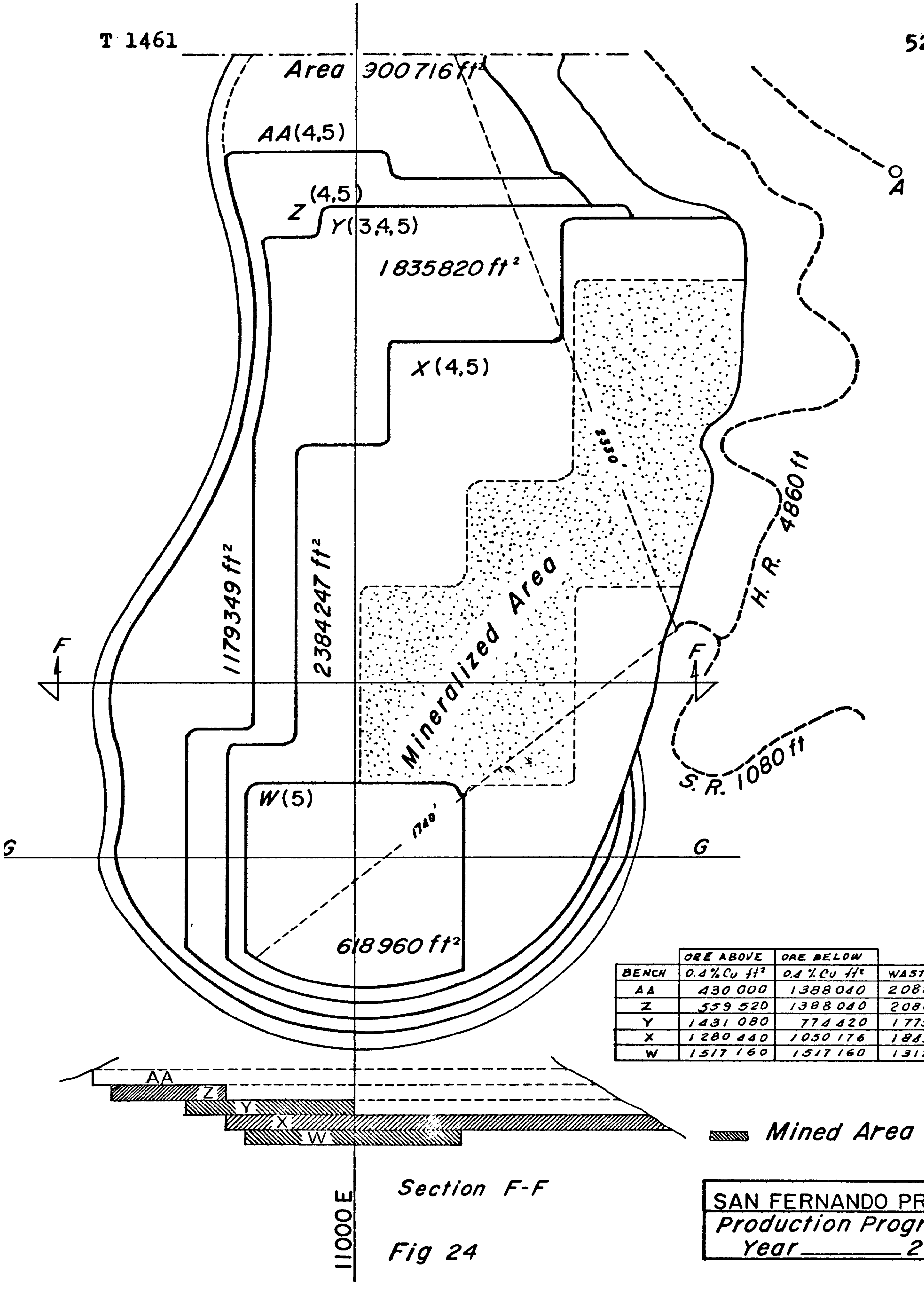


Fig. 23

BENCH	ORE ABOVE 0.4 % Cu ft²	ORE BELOW 0.40 % Cu ft²	WASTE ft²
AA	430 000	1 388 040	2 080 984
Y	1 431 080	774 420	1 775 400
Z	559 520	1 388 040	2 080 984



Section F-F

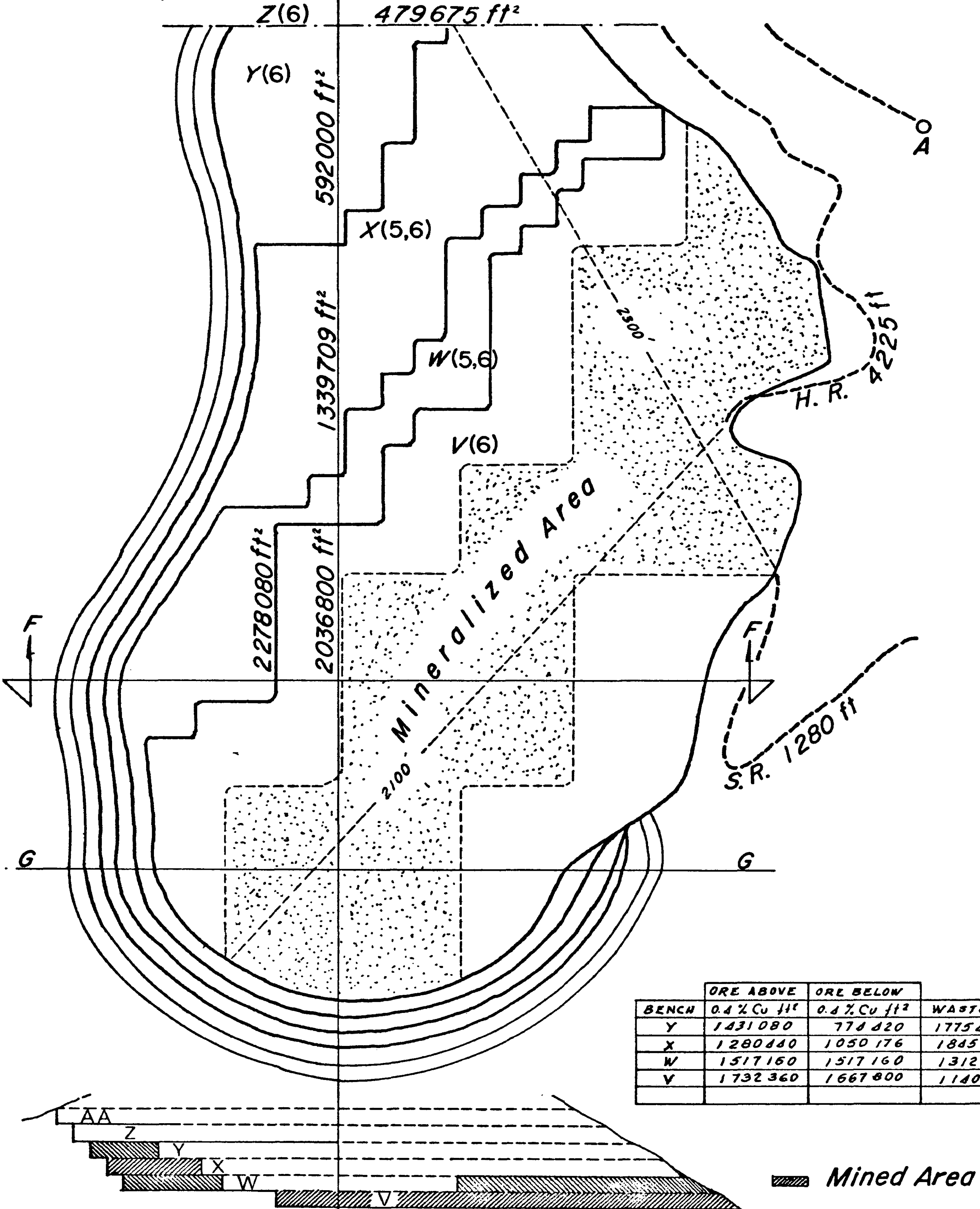
Fig 24

BENCH	ORE ABOVE 0.4% Cu ft²	ORE BELOW 0.4% Cu ft²	WASTE ft²
AA	430 000	1388040	2088984
Z	559 520	1388 040	2080 984
Y	1431 080	774 420	1775 400
X	1280 440	1050 176	1845 340
W	1517 160	1517 160	1312 720

SAN FERNANDO PROJECT

Production Program

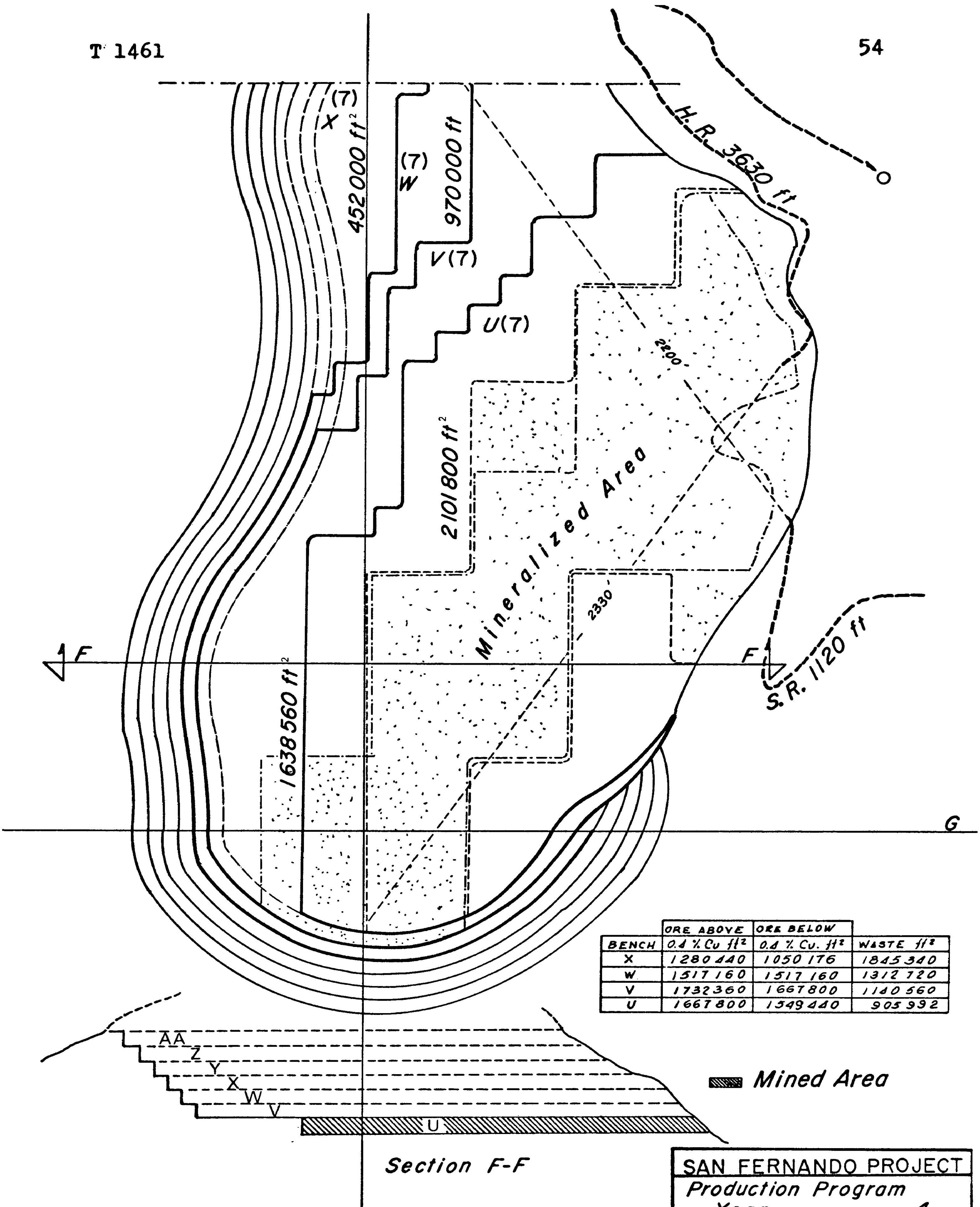
Year 2



BENCH	ORE ABOVE 0.4 % Cu ft²	ORE BELOW 0.4 % Cu ft²	WASTE ft²
Y	1431080	774420	1775400
X	1280440	1050176	1845340
W	1517160	1517160	1312720
V	1732360	1667800	1140560

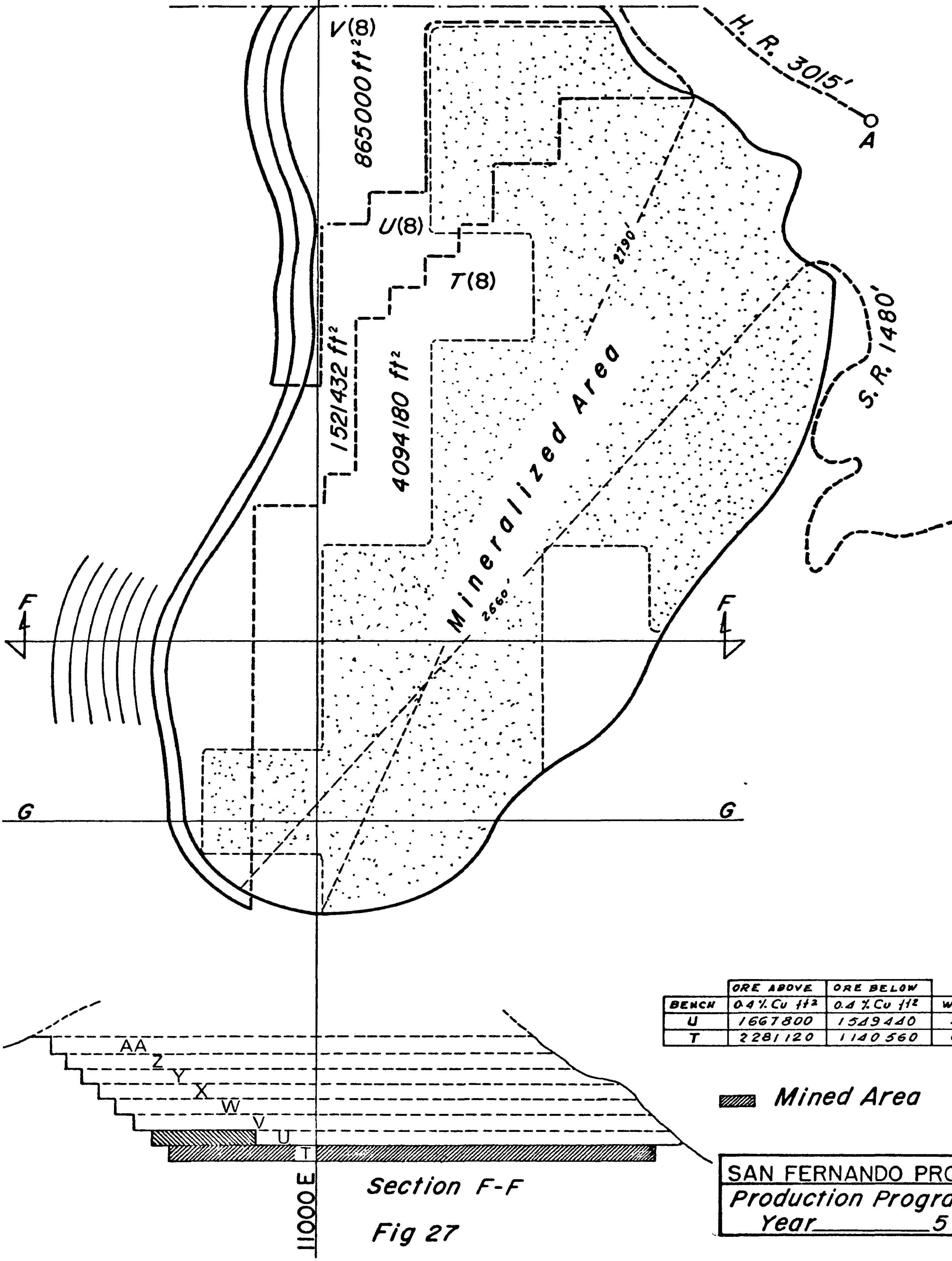
Fig 25

SAN FERNANDO PROJECT
Production Program
Year 3



BENCH	ORE ABOVE 0.4 % Cu ft²	ORE BELOW 0.4 % Cu. ft²	WASTE ft²
X	1280440	1050176	1845340
W	1517160	1517160	1312720
V	1732360	1667800	1140560
U	1667800	1549440	905992

Fig 26

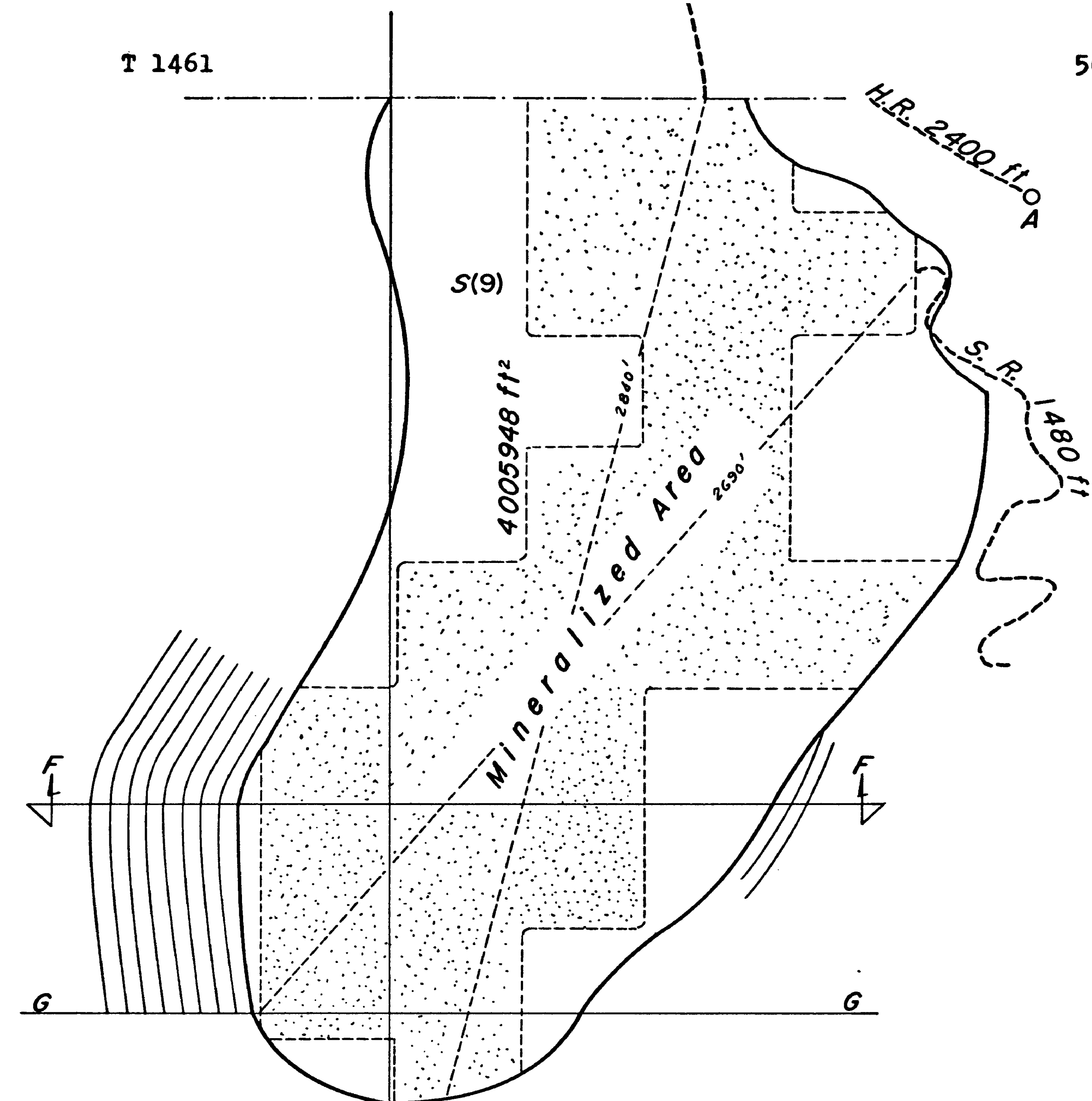


	ORE ABOVE	ORE BELOW	
BENCH	0.4% Cu ft²	0.4 % Cu ft²	WASTE ft²
U	1667800	1549440	905992
T	2281120	1140560	672500

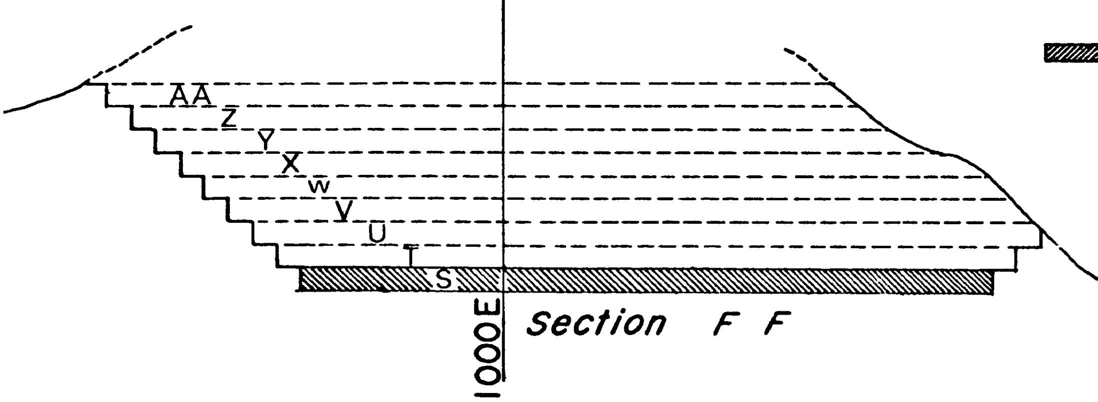
Mined Area

SAN FERNANDO PROJECT
Production Program
Year 5

Fig 27



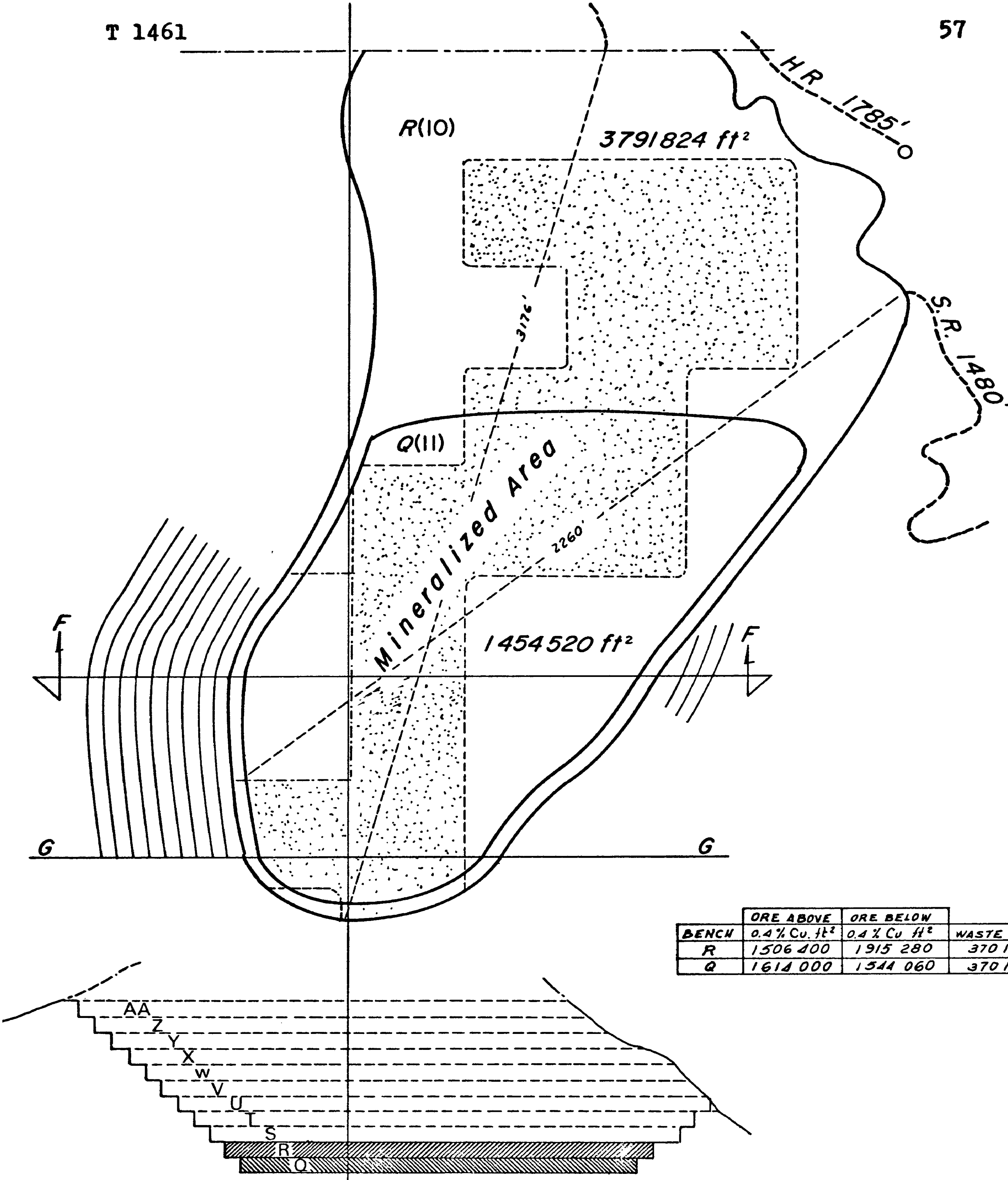
	ORE ABOVE	ORE BELOW	
BENCH	0.4 % Cu ft²	0.4 % Cu ft²	WASTE ft²
S	2 259 600	1 242 780	503 568



Mined Area

Fig 28

SAN FERNANDO PROJECT
Production Program
Year 6

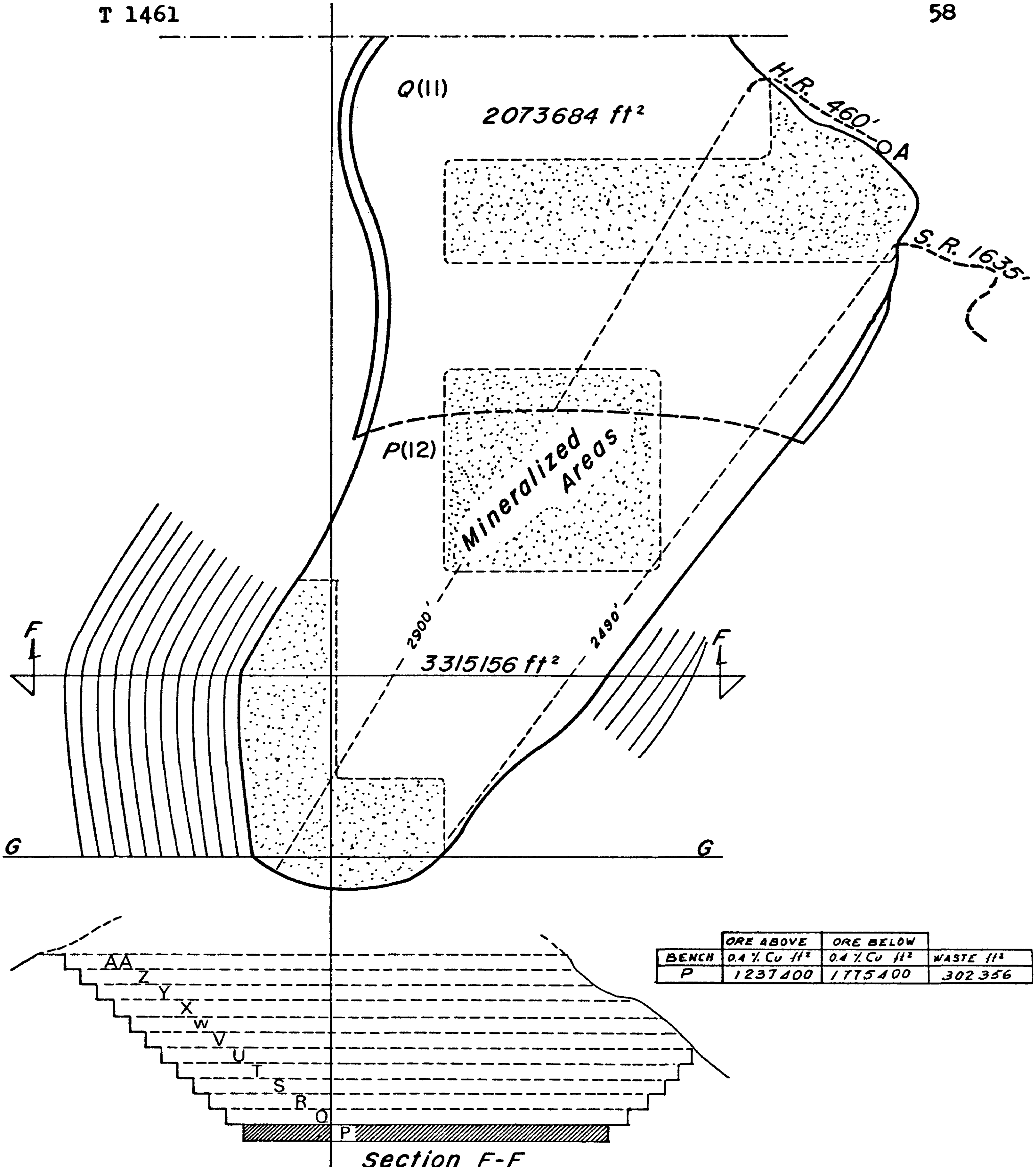


BENCH	ORE ABOVE 0.4% Cu. ft²	ORE BELOW 0.4% Cu ft²	WASTE ft²
R	1506 400	1915 280	370 144
Q	1614 000	1544 060	370 144

Section F-F

Fig 29

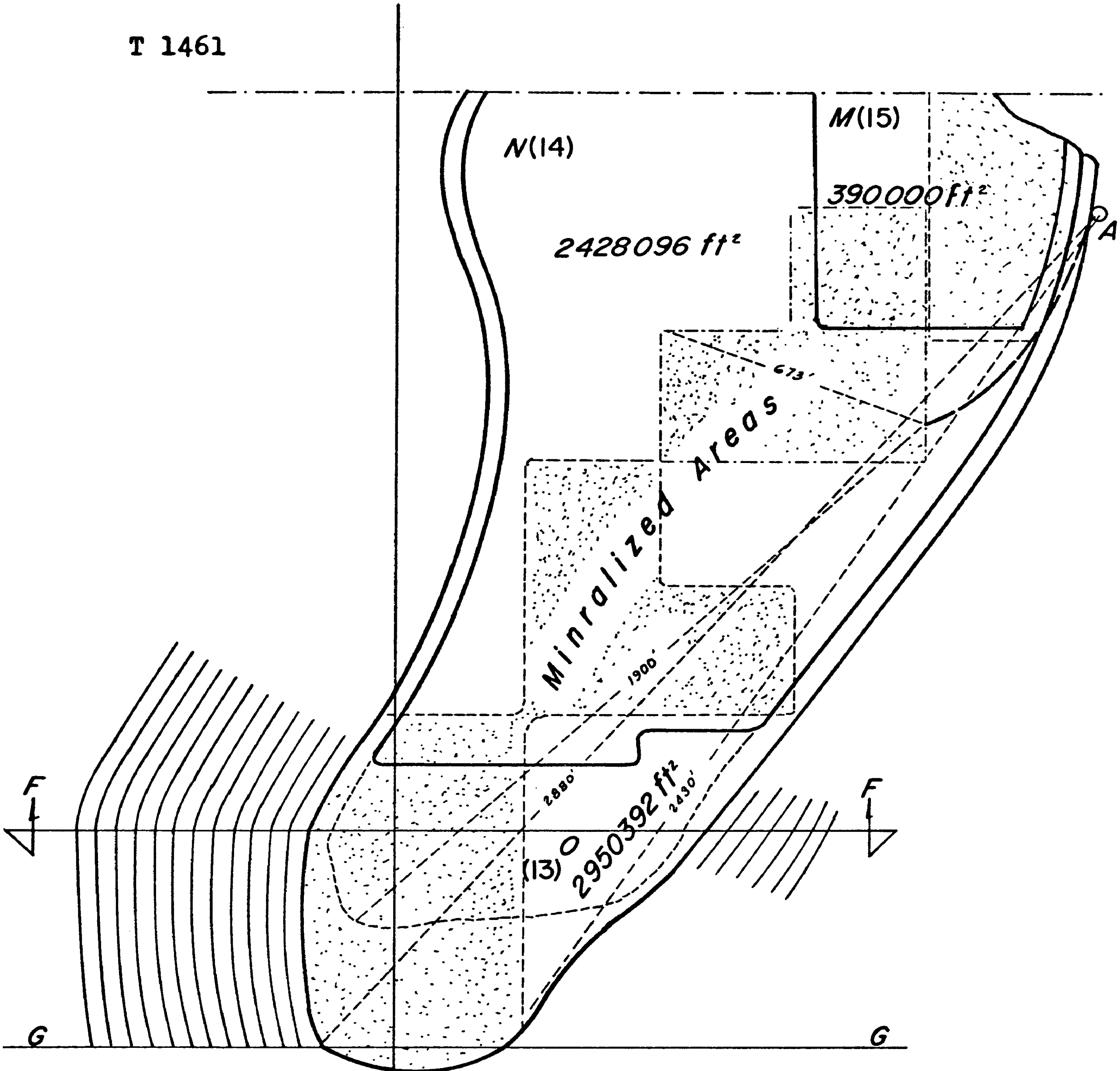
SAN FERNANDO PROJECT
Production Program
Year 7

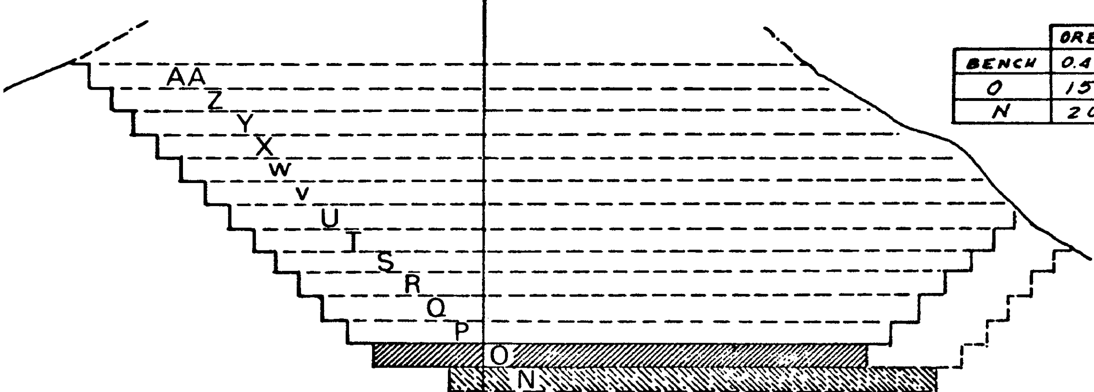


	ORE ABOVE	ORE BELOW	
BENCH	0.4 % Cu ft²	0.4 % Cu ft²	WASTE ft²
P	1237400	1775400	302356

Fig 30

SAN FERNANDO PROJECT
Production Program
Year 8



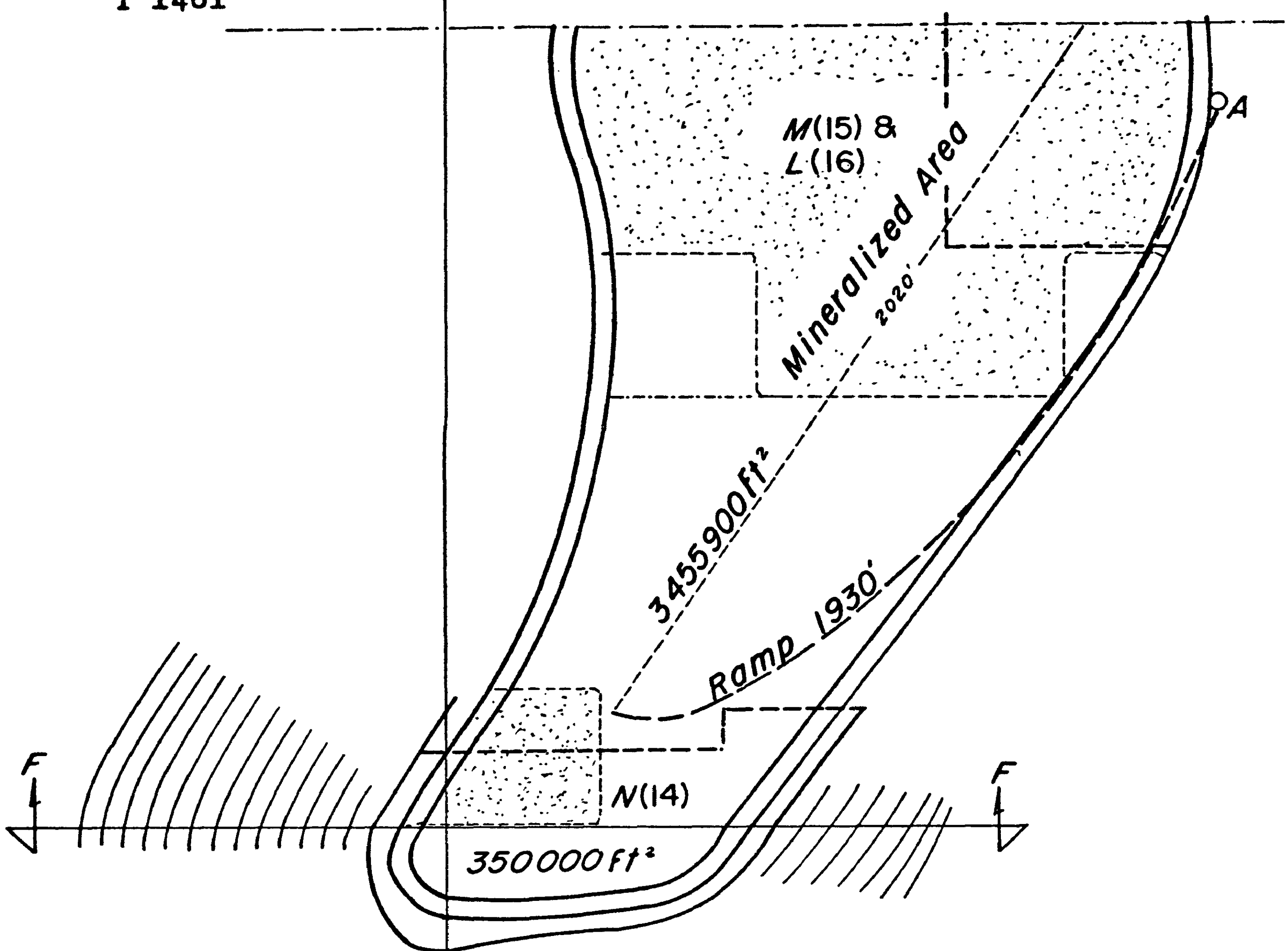


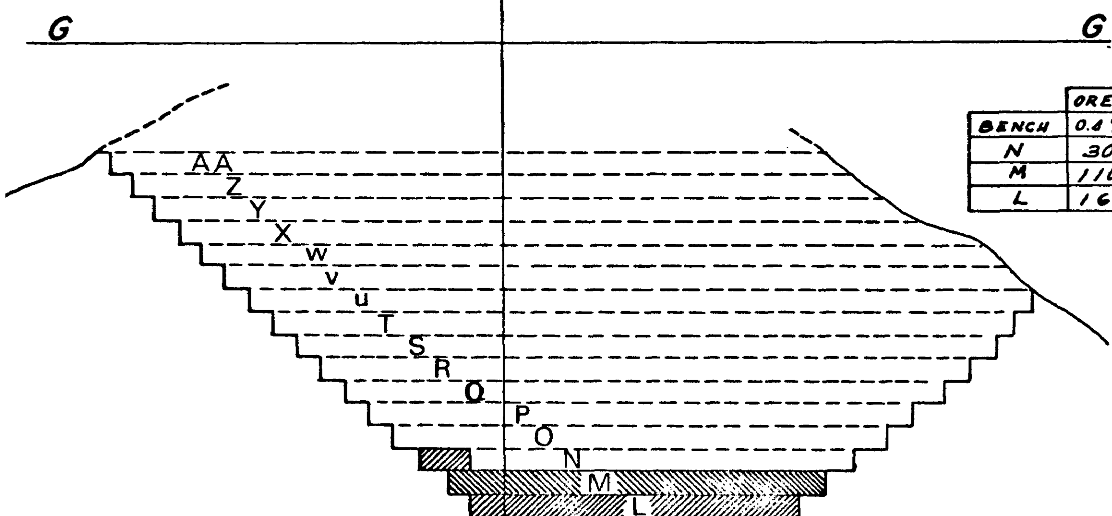
BENCH	ORE BELOW 0.4% Cu ft²	ORE ABOVE 0.4% Cu ft²	WASTE ft²
O	1506400	974856	469136
N	2033640	309888	234568

Section F-F

SAN FERNANDO PROJECT
Product. Program
Year 9

Fig 31



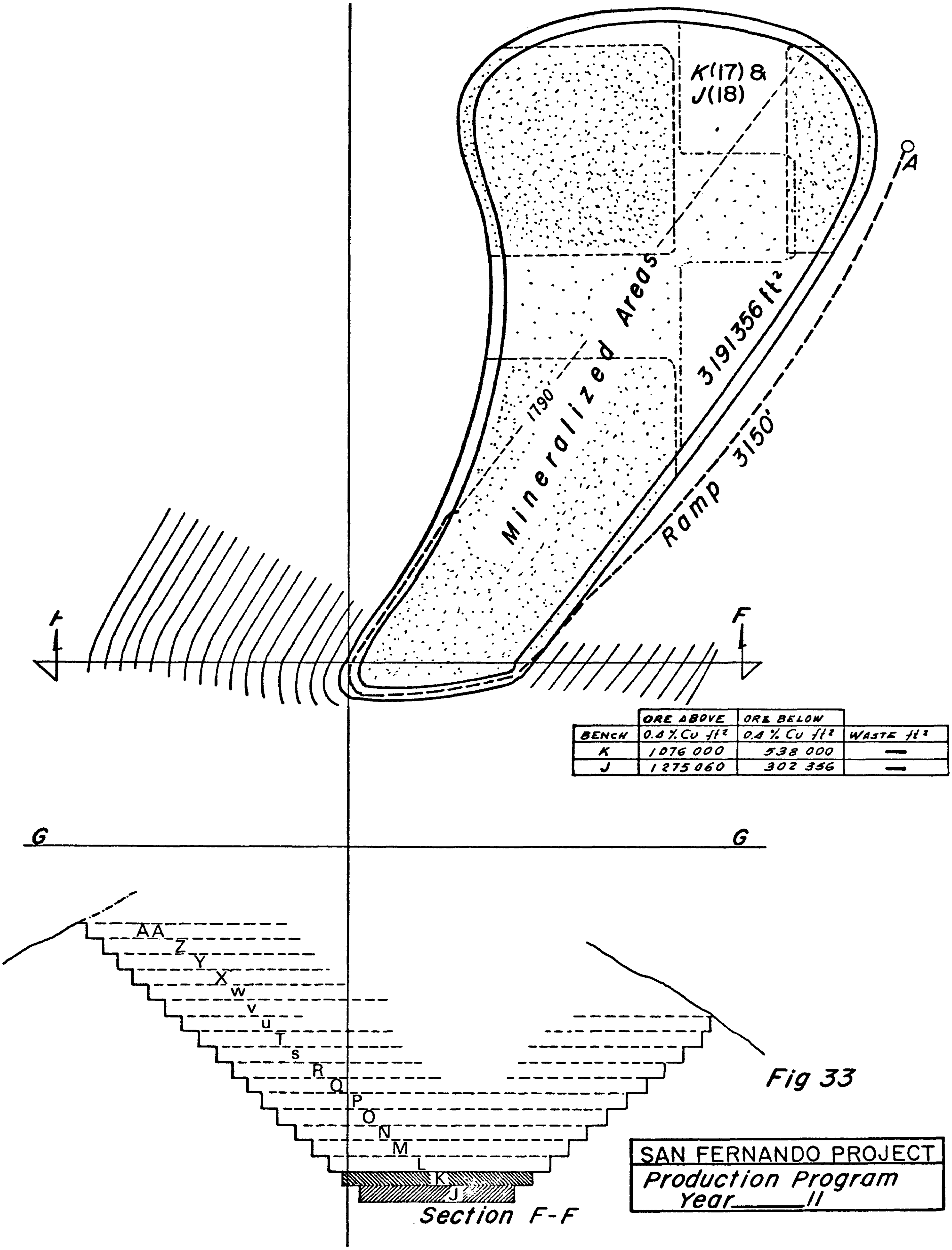


Section F-F

BENCH	ORE ABOVE 0.4% Cu ft²	ORE BELOW 0.4% Cu ft²	WASTE ft²
N	309 888	2 033 640	234 568
M	1 108 280	1 076 000	—
L	1 608 620	269 000	—

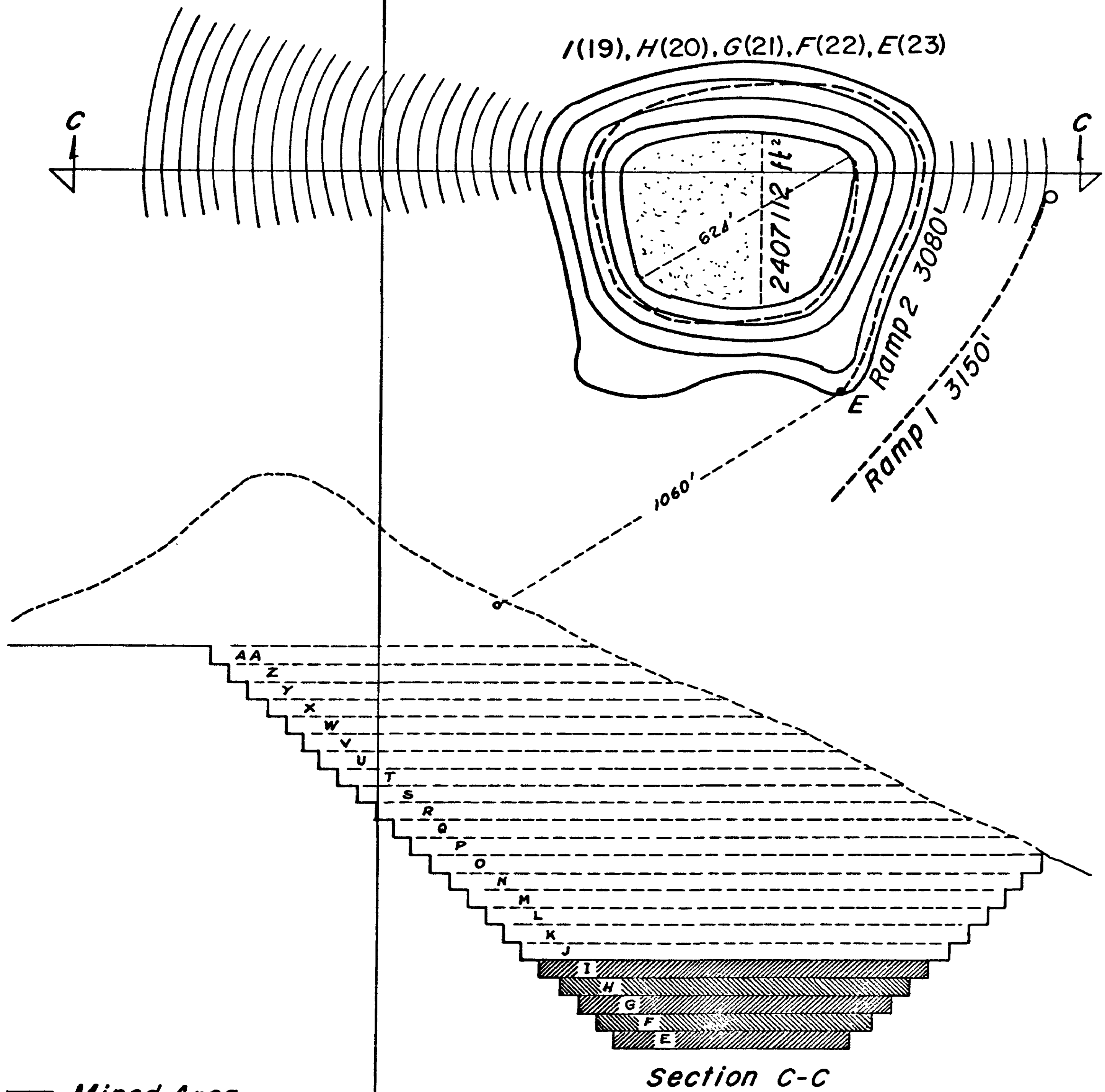
SAN FERNANDO PROJECT
Production Program
Year 10

Fig 32



BENCH	ORE ABOVE 0.4 % Cu ft²	ORE BELOW 0.4 % Cu ft²	WASTE ft²
K	1 076 000	538 000	—
J	1 275 060	302 356	—

Fig 33



Section C-C

BENCH	ORE ABOVE 0.4 % Cu ft²	ORE BELOW 0.4 % Cu ft²	WASTE ft²
I	267 924	459 452	—
H	275 456	275 456	—
G	234 568	194 756	—
F	—	335 712	—
E	251 784	111 904	—

11000E

Fig 34

SAN FERNANDO PROJECT
Production Program
Year 12

Mining Capital Costs

Costs for equipment are taken from available literature, and labor costs are estimated considering actual wage rates for mining labor in Mexico. The valuation of the project is done as a point estimate; that is, without considering probabilistic trends of labor cost, and copper prices for the future. Later, and depending on this first estimate, refinements can be introduced for decision making.

The estimated prices of equipment are given below.

Units	Basic Equipment Description	Price f.o.b. Jobsite
5	40-R Bucyrus Erie drilling machines: Hole size 6-3/4 in. to 9.0 in. Basic price	$ 734,395
3	280-B Bucyrus Erie standard shovel: Nominal dipper capacity; 16 cu yd Effective length of boom; 50 ft Length of dipper handle; 31 ft Basic price; $765,495.00/unit	$2,296,485
2	8-cu-yd Caterpillar loaders: Basic price; $ 91,242.00/unit	182,484
1	5-cu-yd Caterpillar loader: Basic price; $ 91,242.00/unit	91,242
23	R-50 Euclid 50-ton trucks: (19 on production-stripping operations, and 4 on transportation of concentrates). Delivered price; $129,580/unit	$2,980.340
	Auxiliary Equipment	
3	No. 12 Caterpillar motor grader: Estimated price; $81,104/unit	243,312
3	TD-25B Bulldozers: Estimated price; $101,380/unit	304,140
5	1-ton pick-ups: Unitary price; $4,000.00	20,000
2	Portable welding units: Estimated cost; $10,138/unit	20,276

1	45-ton truck-mounted crane: Estimated price	$ 150,000
1	10-ton truck-mounted crane: Estimated price	60,000
1	955 Caterpillar traxcavator: Estimated price	60,000

Mining Operating Costs

Mining operating costs include drilling, blasting, power, loading, hauling, auxiliary equipment, and overhead and clerical.

Drilling. The drilling operating costs include labor, power, maintenance and overhead. The following empirical formulas given by Pfleider (1968, p. 322), are useful for estimating these costs.

$$R_h = 1.5 \times M \times 10^{-4} + 1.25 \times E$$

$$R_f = \frac{R_h}{0.7Z} = \frac{1.5 \times M \times 10^{-4} + 1.25 \times E}{0.7Z}$$

$$O = R_f + B/L$$

where R_h = Rig hourly operating cost in Dlls./hr

M = Machine delivered cost

E = Sum of direct wages of operator and helpers in dollars per hour.

R_f = Rig operating cost per foot of hole in Dlls./ft

Z = Drilling rate in feet per hour. (0.7Z, is the effective drilling rate allowing for 30% down time).

O = Cost of drilling in dollars/ft

B = Cost of drilling bits in dollars

L = Life of bit in feet of hole

Data for these formulas are furnished as follows:

M = $146,879.00

E = $2.00/hr

Z = 36 ft/hr

B = \$1,100.00 (approximate price for 7-3/8 - 7-7/8 in. bits, for medium and hard rocks).

L = 100 hr (this factor is difficult to predict as it varies from 20 to 200 hr according to the abrasiveness of material). For calculation purposes, the life of the bit in feet of hole = 100 hr x 36 ft/hr = = 3,600 ft

Calculations:

$R_h = 1.5 \times \$146,879 \times 10^{-4} + 1.25 \times \$2.00/\text{hr} =$	\$24.53/hr
$R_f = \dfrac{\$24.53/\text{hr}}{0.7 \times 36\ \text{ft/hr}} =$	\$ 0.98/ft
$O = \$0.98/\text{ft} + \$1,100/3,600\ \text{ft} =$	\$ 1.29/ft
Tons per foot of hole =	41
<u>Cost of drilling per ton</u> =	\$ 0.031

<u>Blasting</u>. Calculation of benches has been done in Appendix E. A summary of such data is given below.

Drilling pattern with 7-in. drill holes	22.4 x 22.4 ft
Total drill holes	49
Length of holes	56 ft
Blasting agent	AN/FO
Primers	dynamite 40%
Detonator (electric blasting caps)	
Volume of explosive /hole	8.62 cu ft
Powder factor	0.227 lb/ton

Costs related to a block of 164 x 164 x 164 ft (111,500 ton);

Daily required powder (AN/FO)	23,500 lb
Cost of AN/FO = 23,500 lb x \$0.06/lb =	\$1,410.00
Primers (10-lb,40% dynamite cartridge) = 98 cartridges: \$2.30/cartridge =	296.00

Regular electric blasting caps (two caps per hole)

Delay 0	14 units;	cost	14 x \$0.363	\$5.82
Delay 1	14 "	"	14 x \$0.363	5.82
Delay 2	14 "	"	14 x \$0.363	5.82
Delay 3	14 "	"	14 x \$0.363	5.82
Delay 4	14 "	"	14 x \$0.363	5.82
Delay 5	14 "	"	14 x \$0.363	5.82
Delay 6	14 "	"	14 x \$0.363	5.82
			Total blasting caps. . .	\$40.74

Total blasting materials per 111,500-ton block \$1,747.00

Cost of explosives = \$1,747/111,500 ton = \$0.016/ton

Power. The distribution of electric power is as follows:

Total hp consumed in metallurgical plant = 12,406 (approximately 0.414 hp per ton of ore treated)	9,270 kw	61.8%
Shovels power requirements Appendix B	2,700 kw	18.0%
Drilling rigs power requirements Appendix B	1,000 kw	6.6%
Other	2,030 kw	13.6%
Total basic demand . . .	15,000 kw	100.0%

Costs:

Monthly fixed charges = 15,000 kw x \$0.96/kw =	\$14,400
Monthly kwhr = 15,000 kw x 720 hr/month = = 10,800,000	
90 x 15,000 = 1,350,000 kwhr x \$0.016/kwhr	21,600
180 x 15,000 = 2,700,000 kwhr x \$0.0128/kwhr	34,560
Difference = 6,750,000 kwhr x \$0.0088/kwhr	59,400
Total power costs per month . . .	\$129,960

$$\text{Power milling} = \frac{\$129,960 \times 0.618}{30,000 \times 26 \text{ days}} = \$0.103/\text{ton of ore}$$

$$\text{Power loading} = \frac{\$129,960 \times 0.180}{111,500 \times 26 \text{ days}} = \$0.0081/\text{ton loaded}$$

$$\text{Power drilling} = \frac{\$129,960 \times 0.066}{111,500 \times 26 \text{ days}} = \$0.0022/\text{ton drilled}$$

$$\text{Power other} = \frac{129,960 \times 0.136}{30,000 \times 26 \text{ days}} = \$0.0230/\text{ton of ore}$$

By referring to tons of processed ore:

Power cost per ton of ore (milling)	$0.103
" " " " " " (loading)	0.031
" " " " " " (drilling)	0.008
" " " " " " (other)	0.023
Overall cost of power per ton of ore	$0.165

Loading. This part includes the calculation of ownership and operating costs for both; power shovels and loaders.

Power shovels:

Delivered price	$765,495
Annual depreciation = $765,495/16 years = = $ 47,843/year	
Hourly depreciation = $ 47,843/8,760 hr per yr	$5.46/hr
Average investment = 0.5 x $765,496 = $382,747	
Int., insurance and taxes (average investment x 13%) = $382,747 x 0.13/8,760 =	5.67/hr
Ownership cost excluding depreciation	$5.67/hr
Energy consumption = $0.0081/loaded ton x - 1,374 ton/hr =	$11.10/hr
Lubricants (25% of energy cost) =	2.78/hr
Repair parts (90% of hourly depreciation) =	4.91/hr
Total operating costs excluding labor	$18.79/hr
Total ownership and operating costs- excluding depreciation and labor - $24.46/hr / 1,374 tons per hr	$ 0.0178/ton

Loaders:

Two 8-cu-yd loaders, one 5-cu-yd	
Delivered price	$273,726
Annual depreciation = $273,726/4 yr = = $ 68,431/yr	
Hourly depreciation = $ 68,431/8,760 hr per yr =	$7.80/hr
Average investment = 0.5 x $273,726 = $136,863	
Int., insurance, and taxes (average investment x 13%) = $136,863 x 0.13/8,760 =	2.03/hr
Ownership cost excluding depreciation	$2.03/hr
Power cost. Fuel 30 gal/hr, @ $0.16/gal	$4.80/hr
Lubricants (25% fuel cost)	1.20/hr
Repair parts (50% of hourly depreciation)	3.90/hr
Tires ($4,000/6,000 hr)	0.67/hr
Total operating costs	$10.57/hr
Total ownership and operating costs-excluding depreciation and labor-$12.60/hr/524 tons per hr	$0.0240/ton

<u>Hauling</u>

50-ton truck - delivered price	$ 129,580
Annual depreciation = $129,580/4 yr = = $ 32,395/yr	
Hourly depreciation = $ 32,395/8,760 hr per yr =	$3.69/hr
Average investment = $ 64,790	
Int., insurance, and taxes = = $64,790 x 0.13/8,760 =	0.96/hr
Ownership cost excluding depreciation	$0.96/hr
Power cost. Fuel 16 gal/hr, @ $0.16/gal	$2.56/hr
Lubricants (25% fuel)	0.64/hr
Repair parts(40% of hourly depreciation)	1.48/hr
Tires (Rock tread, front 2, $2,090, extra tread, rear 4, $5,372) Total tires = $7,462. Life = 6,000 hr $7,462/6,000 =	1.25/hr
Total operating costs	$5.93/hr
Total ownership and operating costs-excluding depreciation and labor-$6.89/hr/273 ton per hr	$0.0250/ton

Auxiliary Equipment. This part includes the auxiliary equipment which is to be depreciated in four years. Cranes and welding units are considered by separate because this equipment is depreciable on a 16-year basis.

Units considered in this part are: graders, bulldozers, pick-ups, and traxcavators.

Delivered price	$ 627,452
Annual depreciation = $627,452/4 years = = $156,863/year	
Hourly depreciation = $156,863/8,760 hr per yr	$17.90/hr
Average investment = 0.5 x $627,452 = $313,726	
Int., insurance, and taxes (average investment- x 13%) = $313,726.00 x 0.13/8,760 =	4.65/hr
Ownership cost excluding depreciation	$ 4.65/hr
Power cost (30 gal fuel/hr @ $0.16/gal) =	$ 4.80/hr
Lubricants (25% of fuel cost) =	1.20/hr
Repair parts (40% of hourly depreciation) =	7.16/hr
Total operating costs =	$13.16/hr
Total ownership and operating costs- excluding depreciation and labor- $17.81/hr/4,660 tons per hr =	$0.0038/ton

Cranes and welding units:

Delivered price	$ 230,276
Annual depreciation = $230,276/16 years = = $ 14,400/year	
Hourly depreciation = $ 14,400/8,760 hr per yr =	$ 1.64/hr
Average investment = 0.5 x $230,276 = $115,138	
Int., insurance, and taxes (average investment- x 13%) = $115,138 x 0.13/8,760 =	1.71/hr
Ownership cost excluding depreciation =	$ 1.71/hr
Power cost (10 gal fuel/hr @ $0.16/gal) =	$ 1.60/hr
Lubricants =	0.50/hr
Repair parts (50% of hourly depreciation) =	0.82/hr
Total operating costs =	$ 2.92/hr
Total ownership and operating costs- excluding depreciation and labor- $4.63/hr/4,460 tons per hr =	$0.0010/ton

Total operating costs of auxiliary equipment $0.0048/ton

Overhead and Clerical:

President	@	$ 50,000/yr	$ 50,000.00
V.P. Mining	@	$ 35,000/yr	35,000.00
V.P. Milling	@	$ 35,000/yr	35,000.00
Treasurer	@	$ 30,000/yr	30,000.00
Mine Supt.	@	$ 15,000/yr	15,000.00
1 Accountant	@	$ 8,000/yr	8,000.00
6 Mining Engr.	@	$ 10,000/yr	60,000.00
2 Geologists	@	$ 10,000/yr	20,000.00
3 Surveyors	@	$ 8,000/yr	24,000.00
11 Clerks	@	$ 6,000/yr	66,000.00
5 Secretaries	@	$ 6,000/yr	30,000.00
		Total	$373,000.00

Daily men on payroll 382
Average salary per man-shift = $5.00
Total salaries per year = 382 men x
$5.00/man-shift x 365 days/yr = $697,150.00

Total wages and salaries . . $1,070,150.00

Fringe benefits (Social Security, insurance, etc.), 60% of salaries.
0.60 x $1,070,150.00 $ 642,090.00

Grand total wages and salaries. . . . $1,712,240.00

$$\text{Overhead/ton} = \frac{1{,}712{,}240.00}{111{,}500 \times 300} = \qquad \$0.0510$$

Summary of Stripping Costs/ton of Overburden

Drilling	$ 0.031/ton
Blasting	0.016/ton
Power (included in operating costs of equipment)	-
Loading (0.0178 + 0.0240)	0.042/ton
Hauling	0.025/ton
Auxiliaries	0.005/ton
Overhead	0.051/ton
Total stripping cost	$0.170/ton

Summary of Production Costs Including Stripping Costs

Drilling = $\frac{\$0.031/\text{ton} \times 96{,}000 \text{ tpd}}{30{,}000 \text{ tpd}}$ =	$0.100/ton	of ore
Blasting = $\frac{\$0.016/\text{ton} \times 96{,}000 \text{ tpd}}{30{,}000 \text{ tpd}}$ =	$0.051/ton	" "
Loading =	$0.153/ton	" "
Hauling (Table 12, Appendix C)	$0.124/ton	" "
Overhead	$0.056/ton	" "
Total production cost	$0.484/ton	" "
Unaccountables (10% of production-costs)	$0.048/ton	" "
Grand total production cost	$0.532/ton	" "

Milling Capital and Operating Costs

A summary of estimated costs for the metallurgical plant is shown below. The total cost amounts to $30,282,280 (approximately $1,000 per ton of daily capacity), and the operating cost is estimated at $0.663/ton. The outline for the metallurgical plant is shown in Fig. 35, and the distribution of projected roads in the area is shown in the same figure.

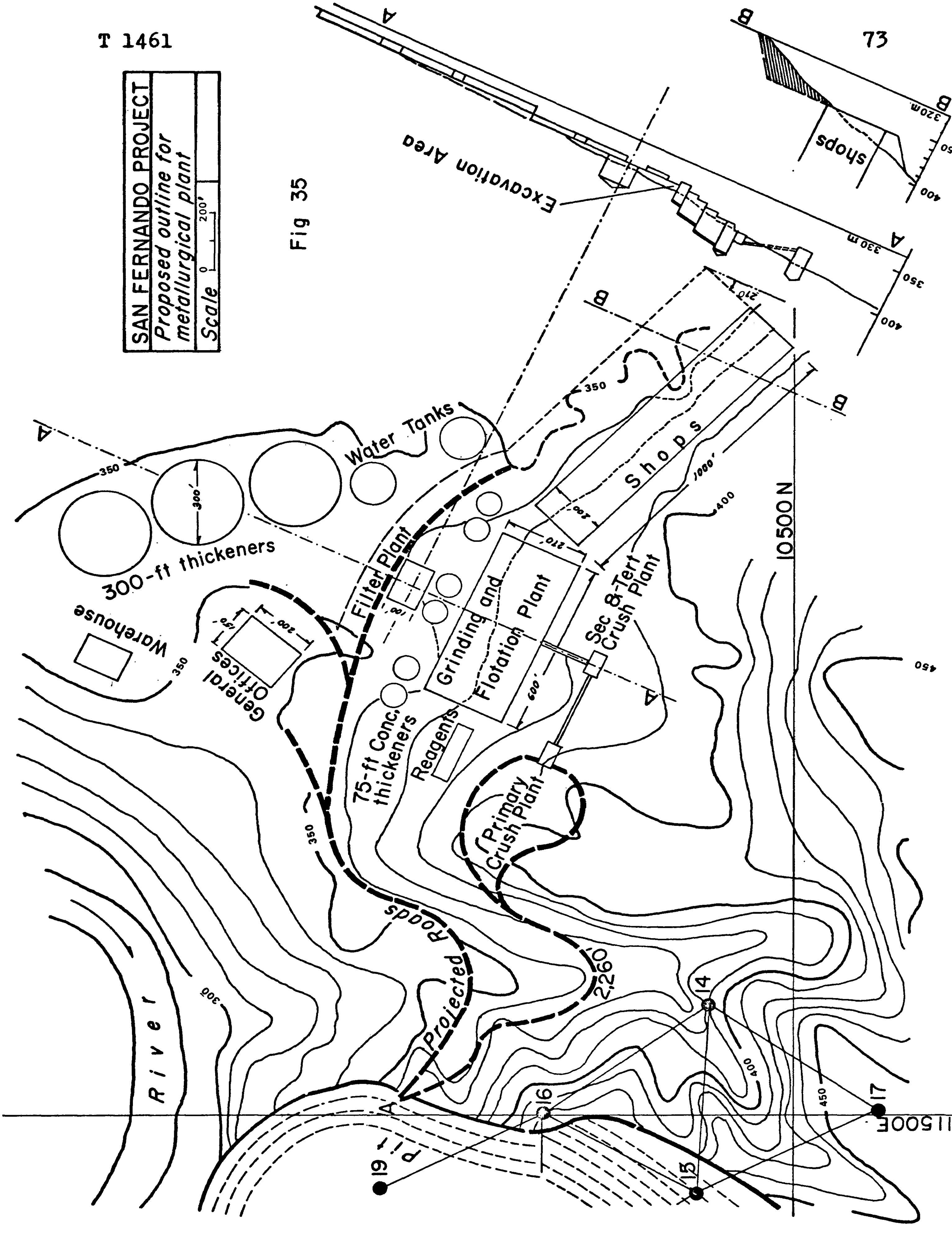
Cost Estimate Summary of a 30,000-tpd Copper Mill

Item	Description		Estimated Erected Cost
1	Site preparation and excavation		$ 218,840
2	Concrete work		2,323,000
3	Buildings		1,892,340
4	Machinery and equipment:		
	Price f.o.b. jobsite =	$9,926,100	
	Installation =	1,050,000	10,976,100
5	Piping		120,000

6	Electrical, excluding motors which are included in machinery	$ 3,890,000
7	Painting	58,000
	Total	$19,478,280
8	Field construction expense (field supervision, clerical, warehousing, recruiting, miscellaneous supplies)	$ 745,000
9	Engineering (planning and design)	2,000,000
10	Housing facilities (462 milling men)	3,100,000
11	Plant start-up expense	30,000
12	Instrumentation	429,000
13	Construction and clean-up	20,000
14	Temporary construction facilities	200,000
15	Construction equipment and accessories	1,000,000
	Total direct construction cost	$27,002,280
	Project management and miscellaneous overhead cost	$ 179,600
	Contingency allowance	2,800,000
	Subtotal of indirect construction costs	$ 2,979,600
	Total of direct and indirect construction costs	$29,981,880
	Miscellaneous and local taxes	300,400
	Total construction cost	$30,282,280

Summary of operating costs

Mining	$ 0.532/ton
Milling	0.663/ton
Total	$ 1.195/ton



SAN FERNANDO PROJECT

Proposed outline for metallurgical plant

Scale 0 200'

Fig 35

Preproduction Expenses

Item	Cost	Total
Power Supply		
Linking substation	$ 94,080.00	
Transmission line	1,901,800.00	
15-mva substation	203,760.00	
200 and 900-kva sub-stations	291,800.00	$ 2,491,440.00
Roads	9,650.00	9,650.00
Exploration costs	840,000.00	840,000.00
Stripping costs: $0.170/ton x 134 million tons		22,780,000.00
Total preparation		$26,121,090.00
Equipment		
5 drilling rigs	$ 734,395.00	
3 shovels	2,296,485.00	
3 loaders	273,726.00	
23 50-ton trucks	2,980,340.00	
3 bulldozers	304,140.00	
5 pick-up	20,000.00	
2 welding units	20,276.00	
2 cranes	210,000.00	
1 traxcavator	60,000.00	
3 graders	243,312.00	
Total equipment	$7,142,674.00	$ 7,142,674.00
Housing: (382 mining men plus 22 concentrate-truck drivers) x $6,500/house		2,620,000.00
Total preproduction expenses mining		$35,883,764.00
Capital cost of metallurgical plant		30,282,280.00
Auxiliary facilities metall. plant		2,777,066.00
Total preproduction expenses milling		$33,059,346.00

Distribution of preproduction expenses

First year;	$35{,}883{,}764 + \frac{1}{4} \times 33{,}059{,}346 =$	\$44,148,600.50
Second year;	$\frac{1}{4} \times 33{,}059{,}346 =$	8,264,836.50
Third year;	$\frac{1}{4} \times 33{,}059{,}346 =$	8,264,836.50
Fourth year;	$\frac{1}{4} \times 33{,}059{,}346 =$	8,264,836.50
	Total preproduction expenses =	\$68,943,110.00
Estimated working capital (W.C.) =		400,000.00

Smelter Returns

A typical smelter schedule is attached to this section. The smelter returns on the copper concentrate are calculated on the basis of this schedule. The considered deduction for copper payment is estimated as 10 kg of copper content per ton of concentrate.

Metallurgical Statement

Ore and Products	Tons Per Day	Analysis Cu%	Recovery Cu%
Feed	30,000	0.53	-
Cu Conc.	523	28	92
Tailing	29,477	0.085	8

Ratio of concentration 57.5

Copper Quotation

New York	51¢/lb
" "	112.455¢/kg
Discount	13.046¢/kg
Difference	99.409¢/kg

Cu; 254 kg - 10 kg = 244 kg @ 99.409¢ = \$242.56

Total payments \$242.56

Smelting charges:

Treatment		\$21.03/ton
Additional charges to treatment		\$ 2.50/ton
Total		\$23.53/ton
Production tax @ \$1.9091 pesos/kg of copper content 244 kg x \$1.9091 = \$465.82 pesos		\$37.26/ton
Exportation tax @ \$3.0631 pesos/kg of copper content 244 kg x \$3.0631 = \$747.43 pesos		\$51.79/ton
Total		\$89.05/ton
50% reduction of production and exportation taxes (Mexicanization)		\$44.58/ton

Freight (truck and rail)

Truck (calculations Appendix D) 74 miles x \$0.017/ton-mile =	\$1.26	
Rail; 372 miles x \$0.016/ton-mile	\$5.95	\$ 7.21/ton
Total deductions		\$75.32/ton
Total payment		\$242.56/ton
Deductions		\$ 75.32/ton
<u>Net return</u>		\$167.24/ton

SMELTER SCHEDULE
COPPER CONCENTRATES

Effective: April 26, 1971

American Currency (Payments)	Weights in metric system
Gold:	Paid at $1.06816 per gram. This payment is based on the quotation of $35.00 an ounce. Any variation goes into the seller's account.
Silver:	From 50 grams up, all the silver content is paid at 95% of the New York's adjusted monthly quotation, adjusted with the London quotation, and discounting 1%.
Lead:	Dry assay (humid assay -1.0 unit). Eighty percent (80%) is paid at the f.o.b. quotation in Monterrey, N.L., with a discount of 10.414¢/kg.
Copper:	Humid assay. Ninety percent (90%) paid with a minimum deduction of 5 kg, and a maximum of 13 kg, with the Mexican quotation, discounting 13.046¢/kg.

Deductions:

Zinc:	Twelve percent (12%) free, excess is paid at 25¢ unit. If it is in excess of 3%, then all zinc is paid at 37.5¢ a unit.
Treatment:	f.o.b. foundry of San Luis Potosi: charged at $21.03 per ton when the price of copper is in excess of 30¢/lb. An addition of 75¢ per each cent of copper price in excess of the previous price is charged up to a maximum of $2.50/ton.

Schedule (cont'd.)

Humidity:	Maximum deduction 1%
Production Tax:	Impure Au, Ag, Cu; Refined Pb.
Exportation Tax:	Refined Ag, and Pb.
Quotations:	Silver: the average of the monthe, lead: the week in which the lots are closed, copper: one week before.
Note:	Charge for treatment based on the rate of exchange at $12.50 pesos per one dollar.

Depletion Allowance

$$\text{Cost depletion} = \$26,121,090 \times \frac{9,000,000 \text{ ton/yr}}{104,000,000 \text{ tons}} = \$2,265,000/\text{yr}$$

Depreciation

The depreciation life of the equipment is summarized below. Part of the equipment is considered to be depreciated in 4 years, and the rest is 16 years. Under these conditions the 4-year depreciable equipment is going to be replaced in the fourth, eighth, and twelfth years of operation.

Equipment	Life
Drilling rigs	4 yr
Shovels	16 "
Loaders	4 "
Bulldozers	4 "
Pick-ups	4 "
Welding units	16 "
Cranes	16 "
Traxcavators	4 "
Trucks	4 "
Mining housing	16 "
Metall. plant and auxiliary facilities	16 "

Depreciable assets:

Total equipment	\$ 7,142,674
Housing	2,620,000
Capital costs of metallurgical plant	30,282,280
Auxiliary facilities	2,777,066
Total.	\$42,822,020
First replacement of 4-year depreciable equipment (fourth year)	\$ 4,615,913
Second replacement in eighth year	\$ 4,615,913

Third replacement in the twelfth year	\$ 4,615,913
Total depreciable investment	\$56,669,759
Annual depreciation \$56,669,759/16 yr	\$ 3,541,860
Total preproduction expense	\$68,943,110
Total distributed replacement costs	\$13,847,739
Grand total distributed investment	\$82,790,849

For the equipment with 4 years of life, the cost of depreciation is capitalized during the 4-year preproduction period. During this period no revenue is obtained at all, and the salvage value of the equipment becomes zero. As consequence a new replacement takes place. A similar situation arises for the equipment with 16 years of life, on which the depreciation period is also capitalized. By taking as zero the beginning of the first year of production, the distribution of the investment, becomes, (Fig. 36).

Year(-4)	0	\$44,148,600.50	\$3,541,860	\$47,690,460.50
Year(-3)	1	\$ 8,264,836.50	\$3,541,860	\$11,806,696.50
Year(-2)	2	\$ 8,264,836.50	\$3,541,860	\$11,806,696.50
Year(-1)	3	\$ 8,264,836.50	\$3,541,860	\$11,806,696.50
		Replacements		
Year(0)	4			\$ 4,615,913.00
Year(4)	8			\$ 4,615,913.00
Year(8)	12			\$ 4,615,913.00
		Working Capital		\$ 400,000.00
		Total capital outlay		\$97,358,289.00

Annual Income Statement

Gross Income	$26,200,000
Operating Costs	10,750,000
Net Income	15,450,000
Less: Depreciation	3,541,000
Depletion	2,265,000
Taxable Income	9,644,000
Federal Income Tax (34% of Taxable Income).	3,275,000
NET PROFIT	$ 6,369,000
Depletion	2,265,000
Depreciation	3,541,000
Annual Cash Flow	$12,175,000

Rate of Return Calculations

The total capital outlay of the project ($97,358,289) is done according to the time chart of Fig. 36. The cash flows are assumed to be uniform during the life of the project. The working capital (W.C.) is estimated as $400,000.

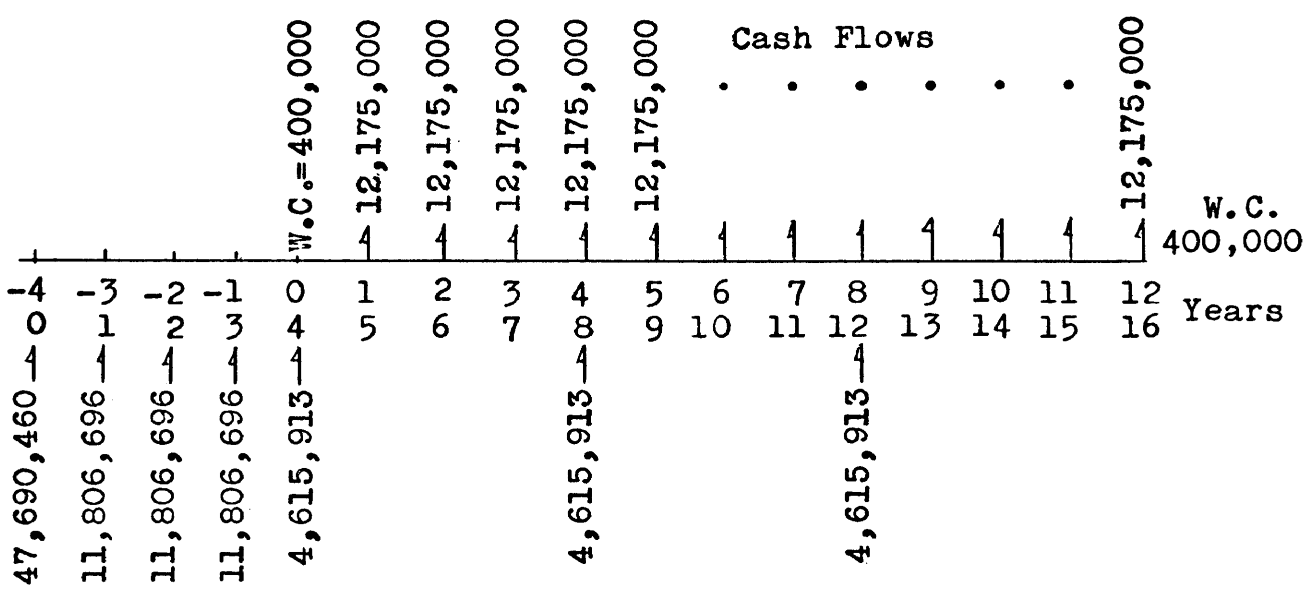


Fig. 36 Rate of return time chart

$$\text{Present value of investment} = 47,690,460(SP_{i-4}) + 11,806,696(SP_{i-3}) + 11,806,696(SP_{i-2}) + 11,806,695(SP_{i-1}) + 4,615,913(PS_{i-4}) + 4,615,913(PS_{i-8}) + 4,615,913 + 400,000$$

$$\text{Present value of cash flows} = 12,175,000(PR_{i-12}) + 400,000(PS_{i-12})$$

On calculation of the rate of return; the present value of the investment is equal to the present value of the cash flows of the project. The factors in the expressions above stands for:

$$SP_{i-n} = (1+i)^n = \text{Single payment compund amount factor}$$

$$PR_{i-n} = \frac{1-(1+i)^n}{i} = \text{Uniform series present value factor}$$

$$PS_{i-n} = \frac{1}{(1+i)^n} = \text{Single payment present value factor}$$

The solution of the above equation by trial and error gives a discounted cash flow rate of return (D.C.F.R.O.R.) = 5%.

Also through the application of the Hoskold formula, it is possible to know the speculative rate of return expected from the investment, by assuming a safe rate on redemption of capital equal to 4%. In this way:

$$P = \frac{A}{i' + \frac{i}{(1+i)^n - 1}}$$

where:

P = Present value of investment

A = Annuity to be purchased, sold or owned

i = Safe rate on redemption of capital

i' = Speculative interest expected from the investment

Present value of the investment for a D.C.F.R.O.R. of 5%;

P =	$109,100,000
A =	12,175,000
i =	4%

By substituting these values in Hoskold's formula; the value of i' becomes 4.7%.

From the results obtained above, it is clear that the project is not feasible under the analyzed conditions.

Sensitivity Analysis

At this point a sensitivity analysis is welcome to have an idea how the different parameters involved in the calculation of the D.C.F.R.O.R. influences this last factor.

The parameters to analyze are:

a) Variations in the initial investment
b) Variations in the life of the project
c) Variations in the cash flows
d) Variations in production costs
e) Effect of subsidization

From these parameters the more important to analyze are; the variation of the initial investment, variations in production expenses, and the effect of subsidization because:

1) The life of the project is assumed fixed (12 years)
2) Variations in the cash flows are assumed to be null because the annual production is fixed, and copper prices are also assumed fixed.

Variations in The Initial Investment. Analysis of variations in the initial investment is important to take into consideration the effect of the discounted cash flow rate of return of a probable error of + 20% in the total investment of the project. The probable error of -20% is not important since it contributes to make the project more infeasible. Results of this analysis are summarized below.

Initial Investment $	Error in Estimate of Investment	D.C.F. R.O.R. %	Variation in D.C.F. R.O.R.
77,500,000	20%	6.5	13%
97,358,289	0	5.0	0

Variations in Production Costs. Variations of +20% in the production costs produces the change in the D.C.F.R.O.R. as stated below.

Production Costs $	Error in Estimate	D.C.F. R.O.R. %	Variation in D.C.F. R.O.R.
8,600,000	20%	5.4	8%
10,750,000	0%	5.0	0

Effect of Subsidization. Subsidization consists in the reduction of approximately 80% in the production and exportation taxes. Under these conditions, the net smelter return is recalculated below.

Total production and exportation taxes	$ 89.05/ton
Subsidized taxes = $89.05 x 20% =	$ 17.80/ton
Freight	$ 7.21/ton
Total treatment	$ 23.53/ton
Total deductions	$ 48.54/ton
Total payment	$242.56/ton
Net smelter return	$194.02/ton

Gross income $30,400,000

Final results are summarized below.

Gross Income	D.C.F. R.O.R.	Variations in D.C.F.R.O.R.
30,400,000	7.5%	15%
26,200,000	5.0%	0

By analyzing the results above it is clear that:

1. The rate of return is influenced more by the subsidization and less by variations in the production costs.
2. Despite the assumed +20% error in the different parameters, the project remains infeasible, on the basis of an acceptable rate of return of 20% for risky enterprises.

APPENDIX A
RESERVES CALCULATION

The drilling network for the San Fernando project covers an area of 98.84 acres (400,000 m^2), involving an orebody which is in the northeast-southwest direction, approximately 3,280 ft (1,000 m) long, and 1,310 ft (400 m) wide. The drilling network was planned to form equilateral triangles with sides 656 ft (200 m) long, but conditions in the terrain did not fit this plan, and the network was deformed to some degree with irregular triangles.

For the calculation of reserves, the body was vertically divided into horizontal slices with a thickness equivalent to a height of benches of 49.2 ft (15 m).

Horizontally the orebody divided with a uniform grid, gave blocks of 328 x 328 ft (100 x 100 m). Interpolation for values on blocks was made horizontally by taking into account the grades of corresponding intercepts of drill holes for a particular bench.

Assay values for each block were estimated by the interpolation method described by Carlson (1966, p. 53). This interpolation method was chosen, because of the convenient way of handling this formula with a computer. The method is flexible, and according to Carlson it gives good correlation between calculations and field results.

Interpolated assays for each block were calculated as shown in Fig. 37. In this figure, five hypothetical drill holes are given inside a circle with a radius of 200 units of length. The center of the given block occupies the center of the circle which covers equal areas and limits the number of drill holes to those that fall within the circle. The radius of the circle is approximately equal to the side of the triangles in the drilling network.

For a given block the drill holes having the most regular angular distribution in relation to the center of the block have been taken. Angles smaller than 30^{o} between any two holes cause the discarding of the farthest hole, because both are in the same direction, and the block is more influenced by the nearest hole.

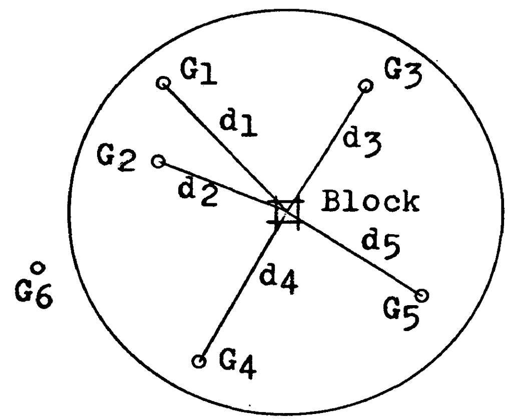


Fig. 37 Calculation of grade for a hypothetical block from the influencing drill holes.

In the figure, the grade for drill hole (1), G(1) is ignored in the calculation of the block's grade, because of its angular displacement with respect to G_2, and G_6 is ignored because it falls out of the circle.

Calculation of the grade for the given block is made through the formula:

$$\text{Grade of block (BLOKG)} = \frac{\sum_{i=1}^{n} G_i \times \left(\frac{1}{d_i}\right)^2}{\sum_{i=1}^{n} \left(\frac{1}{d_i}\right)^2}$$

where G_1 to G_n stand for the assay values of the corresponding drill holes, and d_1 to d_n stand for the distances from the center of the block to the corresponding drill hole.

The FORTRAN program written for the handling of this formula delivers the following data:

1.	BLOCK	Name of block. Fig. 38 shows this heading. The name of the block consists of two parts; the first identifies the bench to which the given block belongs. The second identifies the block itself. (F0025 identifies the block 25 of bench F).
2.	X & Y	Coordinates of the block in meters.
3.	D1-D6	Distances of the holes used in calculations, to the center of the given block.
4.	G1-G6	The grades of holes in the corresponding intercept.
5.	H1-H6	Identification of the holes which enter in calculations.
6.	BLOKG	Computed grade of block.
7.	CUTON	Tons of copper content in each block
8.	J	Number of blocks with computed grade below 0.4% Cu.
9.	K	Number of blocks with computed grade above 0.4% Cu.
10.	CUWGT	Metric tons of copper content from ore below 0.4% Cu.
11.	WGTCU	Metric tons of copper from ore above 0.4% Cu.
12.	SUM	Total metric tons of copper
13.	TONA	Total metric tons of ore with grade below 0.4% Cu.
14.	TONB	Total metric tons of ore with grade above 0.4% Cu.
15.	TTONA	Total of TONA'S in the whole orebody
16.	TTONB	Total of TONB'S in the whole orebody.

The mineralized area for bench F is shown in Fig. 39. In this figure, the program output has been plotted. The rest of the benches is not shown, but the plotting of grades was done on the sections of the orebody, Fig. 2 through 6.

The FORTRAN program is attached to this section, and a summary of the output data follows.

Summary of Reserves (short tons)

Bench	Copper Tons from ore with grade < 0.40%	Ore Tons with grade < 0.40%	Copper Tons from ore with grade > 0.40%	Ore Tons with grade > 0.40%
AA	15,210	5,653,248	12,816	2,609,191
Z	19,035	6,522,979	17,797	3,478,922
Y	14,449	4,783,518	39,259	6,522,979
X	18,728	6,088,113	36,852	6,522,979
W	22,472	7,392,709	35,436	6,522,979
V	24,564	7,827,574	46,795	7,827,574
U	27,950	9,132,170	45,810	7,827,574
T	21,179	6,957,844	60,091	10,871,631
S	24,596	9,132,170	50,268	9,567,035
R	33,512	12,611,092	38,783	6,957,844
Q	30,281	11,741,361	37,194	6,957,844
P	33,971	12,615,501	27,534	5,653,248
O	28,348	11,306,496	20,833	4,348,652
N	37,823	15,655,148	6,730	1,304,596
M	35,512	13,045,957	23,913	4,783,518
L	21,964	9,132,170	44,089	8,697,305
J	24,979	10,001,900	27,291	5,653,248
K	26,642	10,436,766	36,536	7,392,709
I	25,742	10,001,900	10,587	2,174,326
H	22,484	9,567,035	5,658	1,304,596
G	15,516	6,957,844	5,605	1,304,596
F	20,754	8,697,305	3,697	869,730
E	18,207	7,392,709	12,063	2,174,326
D	12,522	5,653,248	11,574	2,174,326
C	10,187	4,783,518	13,863	2,609,191
B	9,172	4,783,518	7,698	1,739,461
A	8,659	3,478,922	5,831	1,304,045
Totals	604,458	231,348,315	685,603	129,154,425

Total number of blocks with grade above 0.40% Cu	295
Sum of grades	155.42
Average grade 155.42/295	0.53% Cu
Ore below 0.40% Cu	536 blocks
Sum of grades	140.24
Average grade 140.24/536	0.26% Cu

```
C     EVALUATION OF AN OREBODY FERNANDO PROJECT
C     DEFINITIUN OF VARIABLES
C     B = BLOCK NAME
C     X AND Y EAST AND NORTH COORDINATES
C     D(1) TO D(N) = DISTANCES FROM HOLE TO CORRESPONDING BLOCK
C     G1 TO G6 = GRADES IN DRILL HOLES
C     H(1) TO H(I) = NUMBERS OF DRILL HOLES IN PLAY
C     N= NUMBER OF TERMS IN CORRESPONDING FORMULA
C     CUWGT = WEIGHT OF COPPER TONS FROM ORE WITH GRADE LT 0.40
C     WGTCU = WEIGHT OF COPPER TONS FROM ORE WITH GRADE GT 0.40
C     M = NUMBER OF BLOCKS WITH GRADE LT 0.40
C     L= NUMBER OF BLOCKS WITH GRADE GT 0.40
C     A = TOTAL TONS OF COPPER FROM ORE WITH GRADE LT 0.40
C     B = TOTAL TONS OF COPPER FROM ORE WITH GRADE GT 0.40
C     TONA = TONS OF ORE A
C     TONB = TONS OF ORE B
C     SUM = TOTAL TONS OF COPPER PER CUT
C     CALCULATIONS
      INTEGER X,Y
      DATA BLKWGT/394500.0/
      TOTWGT=0.
      SUMI=0.
      SUMA=0.
      TTONA=0.
      TTONB=0.
  450 I=0
      J=0
      K=0
      CUWGT=0.
      WGTCU=0.
    5 PRINT 10
   10 FORMAT(///1H1, 9X,5HBLOCK,3X,1HX,5X,1HY,5X,2HD1,4X,2HD2,4X,2HD3,
     1 4X,2HD4,4X,2HD5,4X,2HD6,3X,2HG1,3X,2HG2,3X,2HG3,3X,2HG4,3X,
     2 2HG5,3X,2HG6,2X,2HH1,1X,2HH2,1X,2HH3,1X,2HH4,1X,2HH5,1X,2HH6,
     3 2X,5HBLOKG,1X,7HCU TONS//)
   65 READ 15,B,X,Y,N,D1,D2,D3,D4,D5,D6
   15 FORMAT(A5,I7,I8,I4,6F6.1)
      READ 20,G1,G2,G3,G4,G5,G6,H1,H2,H3,H4,H5,H6
   20 FORMAT(6F7.2,6A3)
      IF(N)320,25,30
   30 GO TO(35,40,45,50,55,60)N
   35 BLOKG=G1
   40 O=(1.0/D1)**2
      P=(1.0/D2)**2
      SNUM=G1*O+G2*P
      SDEN = O+P
      BLOKG=SNUM/SDEN
      IF(BLOKG.LT.0.10)GO TO 65
      WEIGHT=BLOKG*BLKWGT/100.
      PRINT 75,B,X,Y,D1,D2,G1,G2,H1,H2,BLOKG,WEIGHT
   75 FORMAT(10X,A5,2I6,2F6.1,24X,2F5.2,20X,2A3,12X,F7.3,F7.0/)
      IF(BLOKG-0.40)400,800,800
```

```
   45 O=(1.Ø/D1)**2
      P=(1.Ø/D2)**2
      Q=(1.Ø/D3)**2
      SNUM=G1*O+G2*P+G3*Q
      SDEN=O+P+Q
      BLOKG=SNUM/SDEN
      IF(BLOKG.LT.Ø.1Ø)GO TO 65
      WEIGHT=BLOKG*BLKWGT/1ØØ.
      PRINT 9Ø,B,X,Y,D1,D2,D3,G1,G2,G3,H1,H2,H3,BLOKG,WEIGHT
   9Ø FORMAT(1ØX,A5,2I6,3F6.1,18X,3F5.2,15X,3A3,9X,F7.3,F7.Ø/)
      IF(BLOKG-Ø.4Ø)4ØØ,8ØØ,8ØØ
   5Ø O=(1.Ø/D1)**2
      P=(1.Ø/D2)**2
      Q=(1.Ø/D3)**2
      R=(1.Ø/D4)**2
      SNUM=G1*O+G2*P+G3*Q+G4*R
      SDEN=O+P+Q+R
      BLOKG=SNUM/SDEN
      IF(BLOKG.LT.Ø.1Ø)GO TO 65
      WEIGHT=BLOKG*BLKWGT/1ØØ.
      PRINT 95,B,X,Y,D1,D2,D3,D4,G1,G2,G3,G4,H1,H2,H3,H4,BLOKG,WEIGHT
   95 FORMAT(1ØX,A5,2I6,4F6.1,12X,4F5.2,1ØX,4A3,6X,F7.3,F7.Ø/)
      IF(BLOKG-Ø.4Ø)4ØØ,8ØØ,8ØØ
   55 O=(1.Ø/D1)**2
      P=(1.Ø/D2)**2
      Q=(1.Ø/D3)**2
      R=(1.Ø/D4)**2
      S=(1.Ø/D5)**2
      SNUM=G1*O+G2*P+G3*Q+G4*R+G5*S
      SDEN = O+P+Q+R+S
      BLOKG=SNUM/SDEN
      IF(BLOKG.LT.Ø.1Ø)GO TO 65
      WEIGHT=BLOKG*BLKWGT/1ØØ.
      PRINT 1ØØ, B,X,Y,D1,D2,D3,D4,D5,G1,G2,G3,G4,G5,H1,H2,H3,H4,
     1 H5,BLOKG,WEIGHT
  1ØØ FORMAT(1ØX,A5,2I6,5F6.1,6X,5F5.2,5X,5A3,3X,F7.3,F7.Ø/)
      IF(BLOKG-Ø.4Ø)4ØØ,8ØØ,8ØØ
   6Ø O=(1.Ø/D1)**2
      P=(1.Ø/D2)**2
      Q=(1.Ø/D3)**2
      R=(1.Ø/D4)**2
      S=(1.Ø/D5)**2
      T=(1.Ø/D6)**2
      SNUM=G1*O+G2*P+G3*Q+G4*R+G5*S+G6*T
      SDEN=O+P+Q+R+S+T
      BLOKG=SNUM/SDEN
      IF(BLOKG.LT.Ø.1Ø)GO TO 65
      WEIGHT=BLOKG*BLKWGT/1ØØ.
      PRINT 1Ø5,B,X,Y,D1,D2,D3,D4,D5,D6,G1,G2,G3,G4,G5,G6,H1,H2,
     1 H3,H4,H5,H6,BLOKG,WEIGHT
  1Ø5 FORMAT(1ØX,A5,2I6,6F6.1,6F5.2,6A3,F7.3,F7.Ø)
      IF(BLOKG-Ø.4Ø)4ØØ,8ØØ,8ØØ
```

```
  400 CUWGT=CUWGT+WEIGHT
      J=J+1
      GO TO 110
  800 WGTCU=WGTCU+WEIGHT
      K=K+1
      GO TO 110
  110 I=I+1
      IF(I.EQ.25)GO TO 5
      GO TO 65
  320 AJ=J
      BK=K
      SUM=CUWGT+WGTCU
      TONA=AJ*BLKWGT
      TONB=BK*BLKWGT
      PRINT 120,J,K,CUWGT,WGTCU,SUM,TONA,TONB
  120 FORMAT(///30X,4HJ=   ,I2,4X,4HK = ,I2,4X,8HCUWGT = ,F9,0,4X,
     1 8HWGTCU = ,F9.0/1H0,29X,6HSUM = ,F9.0,3X,7HTONA = ,F10.0,3X,
     2 7HTONB = ,F10.0)
      TOTWGT = TOTWGT+SUM
      SUMI=SUMI+CUWGT
      SUMA=SUMA+WGTCU
      TTONA=TTONA+TONA
      TTONB=TTONB+TONB
      GO TO 450
   25 PRINT 125,TOTWGT
  125 FORMAT(//////////40X,23HTOTAL TON OF COPPER = ,F12,0)
      PRINT 130,SUMI
  130 FORMAT(1H0,39X,37HTON COPPER FROM ORE LESS THAN 0.40 = ,F10,0)
      PRINT 140,SUMA
  140 FORMAT(1H0,39X,40HTON COPPER FROM ORE GREATER THAN 0.40 = ,F10.0)
      PRINT 150,TTONA
  150 FORMAT(1H0,39X,31HTOTAL TONS ORE GRADE LT 0.40 = ,F12.0)
      PRINT 160,TTONB
  160 FORMAT(1H0,39X,31HTOTAL TONS ORE GRADE GT 0.40 = ,F12.0)
      GO TO 500
  500 STOP
      END
```

```
3146     1     207620000000     2     177631463146
```

```
ALPHO.  ALPHI.  INTO.  INTI.  FLOUT.  FLIRT.  FLOAT  EXIT
```

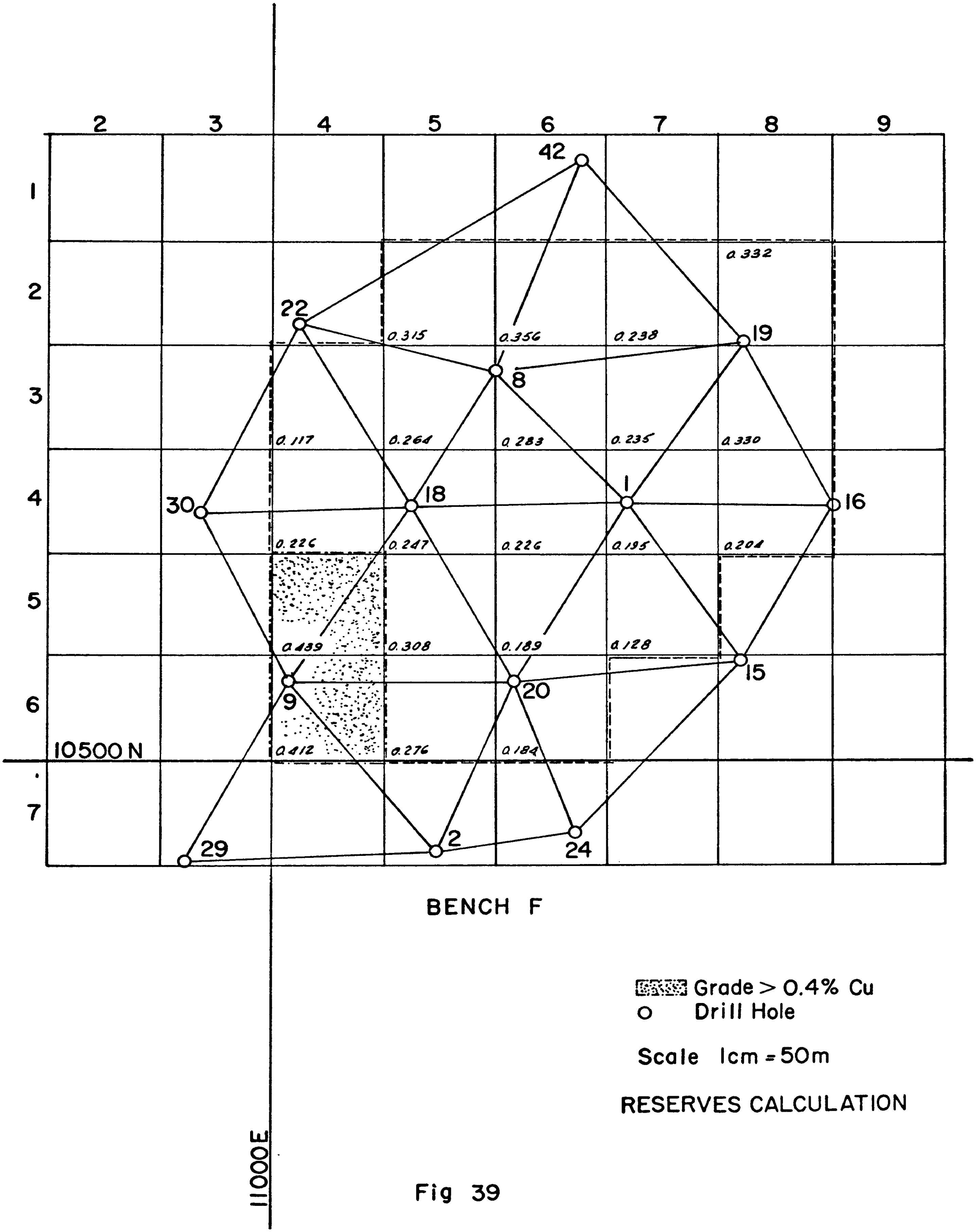
TOTWGT	1240	SUMI	1241	SUMA	1242
I	1245	J	1246	K	1247
B	1252	X	1253	Y	1254
D2	1257	D3	1260	D4	1261
G1	1264	G2	1265	G3	1266

BLOCK	X	Y	D1	D2	D3	D4	D5	D6	G1	G2	G3	G4	G5	G6	H1	H2	H3	H4	H5	H6	BLOKG CU	TONS
F0025	11150	10950	70.0	90.0	130.0				0.40	0.30	0.05				27	8	22				0.315	1241.
F0026	11250	10950	40.0	130.0	210.0	100.0			0.40	0.05	0.20	0.30			27	42	1	8			0.356	1406.
F0027	11350	10950	210.0	170.0	140.0	140.0			0.20	0.30	0.40	0.05			1	8	27	42			0.238	938.
F0028	11450	10950	60.0	100.0					0.38	0.20					19	41					0.332	1311.
F0034	11050	10850	70.0	150.0	130.0	170.0			0.05	0.30	0.25	0.05			22	8	18	30			0.117	462.
F0035	11150	10850	140.0	52.0	200.0	120.0			0.05	0.30	0.20	0.25			22	8	1	18			0.264	1040.
F0036	11250	10850	52.0	125.0	170.0				0.30	0.20	0.25				8	1	18				0.283	1115.
F0037	11350	10850	150.0	110.0					0.30	0.20					8	1					0.235	927.
F0038	11450	10850	60.0	120.0	170.0	250.0			0.38	0.20	0.20	0.30			19	16	1	8			0.330	1301.
F0044	11050	10750	170.0	75.0	180.0	120.0			0.05	0.25	0.68	0.05			22	18	9	30			0.226	891.
F0045	11150	10750	30.0	210.0	130.0	170.0	190.0		0.25	0.05	0.30	0.20	0.25		18	22	8	1	20		0.247	976.
F0046	11250	10750	130.0	70.0	180.0	130.0			0.30	0.20	0.25	0.25			8	1	20	18			0.229	905.
F0047	11350	10750	30.0	170.0					0.20	0.05					1	15					0.195	771.
F0048	11450	10750	130.0	150.0	50.0	160.0			0.20	0.38	0.20	0.05			1	19	16	15			0.204	805.
F0054	11050	10650	140.0	125.0	180.0	80.0			0.05	0.25	0.25	0.68			30	18	20	9			0.439	1730.
F0055	11150	10650	100.0	190.0	100.0	160.0			0.25	0.20	0.25	0.68			18	1	20	9			0.308	1214.
F0056	11250	10650	160.0	120.0	180.0	130.0	80.0		0.25	0.20	0.05	0.05	0.25		18	1	15	5	20		0.189	747.
F0057	11350	10650	100.0	90.0	125.0	150.0	250.0		0.20	0.05	0.05	0.25	0.25		1	15	5	20	18		0.128	504.
F0064	11050	10550	80.0	210.0	170.0	125.0	195.0		0.68	0.25	0.25	0.05	0.05		9	18	20	2	29		0.412	1624.
F0065	11150	10550	130.0	200.0	70.0	170.0	150.0		0.68	0.25	0.25	0.05	0.05		9	18	20	24	23		0.276	1090.
F0066	11250	10550	35.0	210.0	60.0	125.0	125.0		0.25	0.20	0.05	0.05	0.05		20	1	5	24	2		0.184	725.
F0073	10950	10450	140.0	125.0	52.0				0.68	0.05	0.05				9	23	29				0.116	459.

J= 20 K = 2 CUWGT = 18828. WGTCU = 3354.

SUM = 22182. TONA = 7890000. TONB = 789000.

Fig 38 Program Output



BENCH F

Grade > 0.4% Cu
Drill Hole

Scale 1cm = 50m

RESERVES CALCULATION

Fig 39

APPENDIX B

ELECTRIC POWER CONSUMPTION
MINING EQUIPMENT

Shovels

The loading cycle for a 16-cu-yd shovel is shown in Fig. 40. The root mean square hp (Rms hp) for the energy consumption is calculated as 582 kw. The relative calculations for this cycle are made in the assumption of a 90^{o} swing of the shovel. Under this assumption the duration of the cycle is approximately 110 seconds. In the graph, A_1 to A_{17} stand for the values of the areas under the curve. For calculation of this cycle, refer to Neslin and Wright (1971, p.119).

Calculation of the Rms kilowatts.

$A_1 = \frac{733^2}{2} \times 7 =$ 1,880,000 units

$A_2 = 733^2 \times 9.3 =$ 5,000,000 "

$A_3 = \frac{733^2 + 826^2 + 733 \times 826}{3} \times 3.75 =$ 2,280,000 "

$A_4 = 826^2 \times 8.25 =$ 5,610,000 "

$A_5 = \frac{826^2 + 1{,}050^2 + 826 \times 1{,}050}{3} \times 3.0 =$ 2,647,000 "

$A_6 = 1{,}050^2 \times 3 =$ 3,300,000 "

$A_7 = 1{,}050^2 \times 9 =$ 9,900,000 "

$A_8 = \frac{165^2}{2} \times 2.5 =$ 34,000 "

$A_9 = 165^2 \times 9 =$ 245,000 "

$A_{10} = 165^2 =$ 27,200 "

$A_{11} = 276^2/2 \times 3.5 =$ 133,000 "

$A_{12} = \frac{276^2 + 441^2 + 276 \times 4.41}{3} \times 10.25 =$ 1,342,000 "

$A_{13} = 441^2 \times 4.2 =$ 819,000 "

$A_{14} = \frac{441^2}{2} \times 5 =$	489,000 units
$A_{15} = \frac{662^2}{2} \times 5 =$	1,100,000 "
$A_{16} = \frac{662^2}{1} \times 3.25 =$	1,420,000 "
$A_{17} = 662^2 \times 4/2 =$	874,000 "
Total sum	37,100,200 "

$$\text{Rms kw} = \sqrt{\frac{\sum \text{Areas}}{\sum \text{Times}}} = \sqrt{\frac{37{,}100{,}200}{110}} = 582 \text{ kw}$$

As it can be seen from the graph, the maximum peak of energy reaches 1,050 kw, and the calculation of the substations for feeding this equipment should be done considering this peak value.

On consideration of 50 extra transformer kw, the maximum peak value becomes 1,200 kw. Transformers designed for cyclic loads are rated at 72% of maximum peak value; this means that for a maximum peak of 1,200 kw, the transformer should be rated at $1{,}200 \times 0.72 = 864 \approx 900$ kva. For shovel requirements :

Portable substations	3 units	
Capacity (one unit)	900 kva	
Transf. relation	13.8/2.40 kv	
Total power consumption (three shovels)		2,700 kva
Drilling rigs:	5 units	
Power consumption (one unit)	250 hp	
250 x 0.746	184 kw	
Portable substation (200 kva)	5 units	
Transf. relation	13.8/2.40 kv	
Total power requirement		1,000 kva
Total power requirement mining equipment		3,700 kva

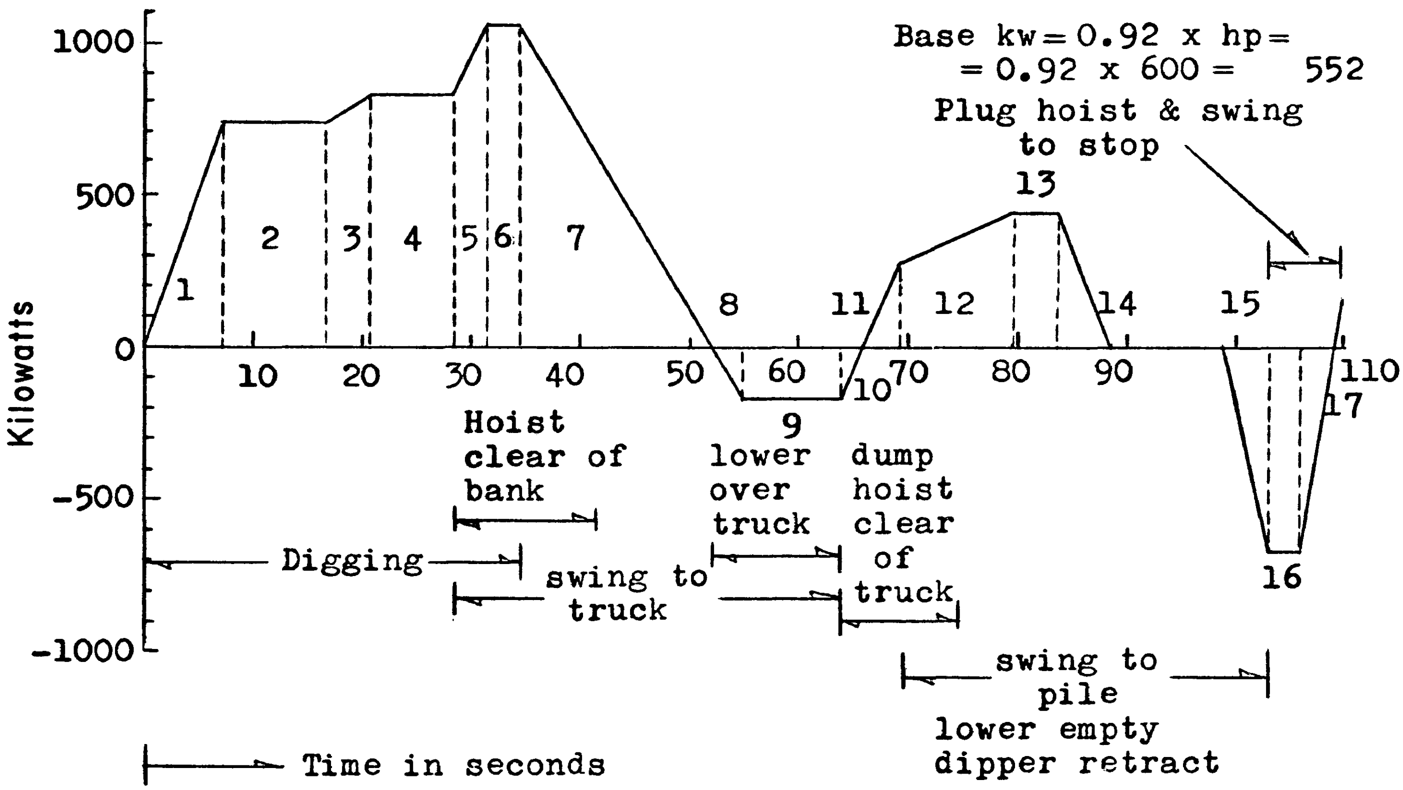


Fig. 40 Working cycle for 16-cu-yd power shovels.

APPENDIX C

CALCULATION OF STRIPPING AND PRODUCTION EQUIPMENT

Loading Units

The basic loading and hauling equipment is considered to consist of 16-cu-yd power shovels, in combination with 50-ton trucks. Factors used in this calculation are given below.

1 bank cubic yard (BCY) = 4,400 lb = 2.20 tons
1 Loose cubic yard (LCY) = 3,310 lb = 1.66 "

The figures from the Power Crane and Shovel Association given below, are used in calculating productivities of equipment (shovels, and loaders), for different bucket capacities. Those figures represent production based on a 90^{o} swing for shovels, from pile to hauler, and in a 60-min hr (for 16-yd bucket). The figures are extrapolated from the corresponding table.

Bucket Capacity (yd)	Easy Digging	Medium Digging	Hard Digging
16	2,290 LCY/hr	1,940	1,710

For calculation, the average of 1,940 and 1,710 LCY/hr (1,820 LCY/hr) was estimated as hourly production.

The following dipper factors are also important in calculation.

Shovel Dipper Factors

Easy Digging	Medium Digging	Hard Digging	Rock
0.85-1.00	0.80-0.90	0.70-0.80	0.40

For calculation, the dipper factor is taken as 0.55

Shovel loading time (S.L.T.) =

$$= \frac{\text{Truck capacity (tons)} \times 2{,}000\ \text{lb/ton} \times 60\ \text{min/hr}}{(\text{LCY/hr}) \times \text{Dip. F.} \times 3{,}310\ \text{lb/LCY}} =$$

$$= \frac{50 \times 2{,}000 \times 60}{1{,}820 \times 0.55 \times 3{,}310} = 1.815\ \text{min} = \qquad 109\ \text{sec}$$

Estimated stripping period = 4 yr

$$\text{Daily stripping} = \frac{133{,}751{,}777\ \text{tons}}{4\ \text{yr} \times 300\ \text{day/yr}} = \qquad 111{,}500\ \text{tons}$$

Hours of regular working = 50 min

Daily effective time = 24 hr x 50/60 = 20 hr

Daily shovel production (50-ton trucks) =

$$= \frac{50\ \text{ton} \times 20\ \text{hr} \times 60\ \text{min/hr}}{1.815\ \text{min}} = \qquad 33{,}000\ \text{tpd}$$

Required shovels = 111,500/33,000 = 3.33

The fraction of shovel can be supplied with one front-end loader calculated below.

Loader Calculation

Loader prod. = (111,500 - 99,000)/20 = 625 ton/hr

Cycle time for medium digging conditions 0.40 min

$$\text{Max. Prod. Rate} = \frac{\text{Bucket LCY} \times 50\ \text{min/hr}}{\text{Cycle time (min)}} \times \frac{\text{Mat'l weight (lb/LCY)}}{2{,}000\ \text{lb/ton}};$$

$$\text{Bucket LCY} = \frac{625\ \text{ton/hr} \times 0.40\ \text{min} \times 2{,}000\ \text{lb/ton}}{50\ \text{min/hr} \times 3{,}310\ \text{lb/LCY}} =$$

$$= 3.02\ \text{LCY} \approx \qquad 5\ \text{LCY}$$

Loader Capacity 5 LCY

With one extra loading unit considered for maintenance purposes, it is possible to choose two 8-cu-yd loaders, and the summary of the loading equipment becomes:

LOADING EQUIPMENT

Three	16-cu-yd	Power shovels
Two	8-cu-yd	Loaders
One	5-cu-yd	Loaders

Stripping Hauling Equipment Calculation

While the number of loading units is constant at any point of the pit, the number of hauling units should vary to keep the loading equipment busy in different parts of the pit. On this basis, the calculation of the hauling equipment is made particularly for each bench.

For hauling equipment calculations the table below summarizes the characteristics of the 50-ton-rear-dump truck manufactured by Euclid.

	Velocity		Gear	
Total Grade (%)	Loaded	Empty	Loaded	Empty
10	10 miles/hr	22 miles/hr	3	5
8	12 " "	25 " "	4	6
6	17 " "	33 " "	4	6
4	23 " "	33 " "	5	6
2	33 " "	33 " "	6	6
-8	18 " "	36 " "	3	6
-6	25 " "	36 " "	5	6
-4	25 " "	36 " "	5	6
-2	36 " "	36 " "	6	6

The waste disposal areas are shown in the surface map (Fig. 1), and in order to have a good equilibrium in the performance of the equipment, it is deemed necessary to operate according to the following assumption in what is considered the total distance of the hauling cycle. The assumption consists in dumping at the farthest distance from point D in Fig. 8 through 21, when the distance of working places to points B or C is minimum. The contrasting part applies; that is, to dump near from point D, when the distance from points B or C to the working places is maximum. Under these conditions, it is possible to have an almost constant distance for the hauling cycle, at any point of the working bench.

For bench CC in Fig. 20;

Distance for hauling cycle = 738 ft (ramp) +
+ (1,560 ft + 10%) dist. hor. dist. =
= 738 + 1,716 = 2,454 ft

Equipment Characteristics. The hauling equipment is assumed to consist of 50-ton trucks.

Struck capacity	33.3 cu yd
Effective capacity = 33.3 LCY x 1.66 ton/LCY =	55.1 tons

For calculation, the rated 50 ton capacity of the vehicle will be used.

Times. Estimated times for the fixed components of the hauling cycle are as follows:

Turning and dumping time average conditions =	1.30 min
Spotting time =	0.40 "

General. The maximum working altitude above sea level is 2,460 ft (748 m). In these conditions, no loss of efficiency of the vehicles is considered in relation to this factor.

Rolling resistance is considered constant as 2%.

Table 8, gives the related factors in the calculation of hauling equipment for bench OO. The number of units obtained from the calculations is 5.1 trucks per loading unit.

Table 8. Calculation of Stripping Hauling Equipment

Bench 00			Loaded					
Road Section	Length Ft	Rolling Resist. %	Percent Grade	Trans. Gear	Max. Speed m.p.h.	Speed Fact.	Average speed m.p.h.	Hauling t. min
1	940	2	0	6	33	0.62	20.5	0.51
2	3,240	2	-8	5	25	0.96	24.0	1.54
				Total hauling time (min) =				2.05
			Empty					
2	3,240	2	8	5	32	1.00	22.0	1.68
1	940	2	0	6	36	0.65	21.4	0.50
				Total return time (min) =				2.18

Loading time (min)	1.82
Turning and dumping time	1.30
Spotting time (min)	0.40
Total time per hauling cycle (min) =	7.75

$$\text{Average trips/hr} = \frac{\text{50 min Prod. hr}}{\text{Total cycle time}} = \frac{60}{7.75} = 6.45$$

$$\text{Hourly tonnage} = \frac{\text{111,500 tpd}}{\text{20 hr/day}} = \quad 5{,}580 \text{ ton}$$

Tons per truck/hr = trips/hr x 50 ton/trip =
= 6.45 x 50 = 322 ton

Hourly prod./loading unit = 5,580 ton/3.38 = 1,645 ton/hr

$$\text{Trucks per loading unit (T.P.L.U.)} = \frac{1{,}645}{322} = 5.1$$

Total trucks bench 00 =
= 5.1 T.P.L.U. x 3.38 load'g units = 17

Table 9 summarizes calculations for the totality of the totality of the stripping benches. In this table, the dumping-

Table 9. Summary of Calculations of Hauling Equipment in Stripping

Bench	Dumping Height	Elevation Diff. ft	Distributed. Horizontal Dist. ft	8% Inclined Dist. ft	% Inclined Distance	Total Haulage Dist. ft	No. of Trucks	Additional Productivity ton/hr	Cumulative Days
OO	2,190	-259	940	3,240	100	4,180	17	0	24
NN	"	-210	1,035	2,630	100	3,665	16	590	52
MM	"	-161	1,085	2,020	100	3,105	14	1,155	84
LL	"	-112	1,100	1,400	100	2,500	13	1,645	121
KK	"	- 62	1,080	780	100	1,860	12	1,645	163
JJ	"	- 13	1,310	163	100	1,473	11	1,645	221
II	"	11	1,460	138	100	1,598	11	1,645	282
HH	"	26	1,550	325	100	1,875	11	1,645	364
HH	2,065	- 43	1,100	2,950	18.2	4.050	16	590	*
II	"	- 92	1,100	2,950	39.0	4.050	15	662	*
JJ	"	-141	904	2,950	59.0	3,854	16	360	*
KK	"	-190	360	2,950	80.5	3,310	16	625	*
LL	"	-239	1,650	3,000	100	4,650	16	254	*
GG	"	7	2,200	2,950	2.4	5.150	16	309	500
FF	1,805	-206	1,826	2,580	100	4,406	17	132	607
EE	"	-157	1,942	1,970	100	3,912	16	309	722
DD	"	-108	1,705	1,355	100	3,060	14	1,190	830
CC	"	- 59	1,716	738	100	2,454	13	1,645	945
BB	"	- 10	1,876	125	100	2,001	12	1,645	1,036
						Total number of days			1,036

* Figures included in the first part of the bench. Fig. 8 to 21

height heading gives the elevation of 3 waste-dumping areas. The elevation difference heading, gives the difference in elevation between the given bench and the corresponding dumping area. The next three columns, give the horizontal and inclined distances considered in the hauling cycle, and the other columns are self-explanatory, except the column for additional productivity which is explained later. Fig. 8, to 21 can help in clarifying the data content of the table.

The size of the fleet has a maximum of 17 and a minimum of eleven 50-ton trucks for stripping operations. From these observations it can be concluded that to avoid having so many idle trucks in benches where less equipment is necessary; the use of an extra loading unit can give to this new arrangement the possibility of removing the overburden in a shorter period. In this respect, the extra loading unit for maintenance purposes can be used at an advantage without any increase in new equipment.

Additional Productivity. The addition of the new unit gives the extra productivity in tons per hour referred before. Calculations for benches 00 and NN are given below.

Bench 00

Productivity 111,500 tpd

Bench NN

Regular Prod. 111,500 tpd

$$\text{Addit. Prod.} = \frac{\text{(17.25 trucks, bench 00 - 15.6 trucks req'd on bench NN) x 1,645 ton/hr per L.U.}}{\text{4.6 trucks/L.U. bench NN}} =$$

$$= \frac{1.6 \times 1{,}645}{4.60} = \quad 590 \text{ ton/hr}$$

590 ton/hr x 20 hr = 11,800 tpd

The value of 590 ton/hr is given in Table 9. For the rest of the benches calculations were done in the same fashion.

Stripping Time

The days consumed in stripping are given in the last column of Table 9 on a cumulative basis, and the calculation for benches OO and NN is shown below.

Bench OO

Total stripping tonnage =		2,720,000 tons
Regular productivity =	111,500 tpd	
Additional Prod. =	0	
Total =	111,500 tpd	

$$\text{Stripping days} = \frac{2{,}720{,}000 \text{ tons}}{111{,}500 \text{ tpd}} \qquad 24 \text{ days}$$

Bench NN

Total stripping tonnage =		2,490,000 tons
Regular productivity =	111,500 tpd	
Additional Prod. =	11,800 "	
Total =	123,300 tpd	

$$\text{Stripping days} = \frac{2{,}490{,}000 \text{ tons}}{111{,}500 \text{ tpd}} = \qquad 28 \text{ days}$$

Days on cumulative basis = 52

Effective time in stripping = 1,036 days

Production Hauling Equipment

The production hauling equipment can be divided into two parts: One is the hauling equipment utilized in the auxiliary stripping operation, and the other is the equipment directly used in production. The hauling equipment calculation is done independently for each one of the two parts. Table 10 shows calculations of hauling equipment for bench AA, and the summary of calculations for the whole operation is given in Table 11. Fig. 23 to 34 can help in clarifying the data in these tables.

Table 10. Calculation of Production Hauling Equipment

Bench AA				Loaded			Production	
Road Section	Length ft	Rolling Resist. %	Percent grade	Trans. Gear	Max. Speed m.p.h.	Speed Factor	Average Speed m.p.h.	Hauling Time min
1	1,920	2	0	6	33	0.78	25.7	0.85
2	7,320	2	-8	5	25	0.96	24.0	3.46
3	2,260	2	-8	5	25	0.96	24.0	1.07
				Total hauling time				5.38
				Empty				
3	2,260	2	8	5	22	1.00	22.0	1.17
2	7,320	2	8	5	22	1.00	22.0	3.78
3	1,920	2	0	6	33	0.90	29.7	0.73
				Total return time				5.68
Average trips/hr = 50/14.58 = 3.43					Loading time			1.82
Tons per hr = 3.43 x 50 = 171.50					Dump, turn, and			
Trucks/load. unit = 1,645/171.5 =					spotting time			1.70
= 9.6 ≈ 10					Total cycle time			14.58
Bench AA				Loaded			Stripping	
1	1,886	2	0	6	33	0.78	25.8	0.83
2	1,150	2	-8	5	25	0.86	21.5	0.61
				Total hauling time				1.44
				Empty				
2	1,150	2	8	5	22	1.00	22.0	0.59
1	1,886	2	0	6	33	0.89	29.0	0.73
					Total return time			1.32
Average trips/hr = 50/6.28 = 7.95					Loading time			1.82
Tons per hr = 7.95 x 50 = 398.00					Dump, turn, and			
Trucks/load. unit = 1,645/398.00 =					spotting time			1.70
= 4.14 = 4; Total = 8 units					Total cycle time			6.28

Production Hauling Costs

Table 11. Summary of Production Hauling Equipment

Bench	Elevation Diff. ft	Distributed Hor. Dist. ft	Inclined Dist.(-) ft	Inclined Dist.(+) ft	Total Haulage Dist. ft	50-ton Trucks	Total Trucks Production & Stripping
Production							
AA	584	1,920	9,580		11,500	10	18
Z	535	2,090	8,965		11,055	9	17
Y	485	1,970	8,350		10,320	9	18
X	436	1,410	7,735		9,145	8	18
W	387	1,740	7,120		8,860	8	17
V	338	2,100	6,485		8,585	8	17
U	288	2,330	5,890		8,520	7	15
T	239	5,275	5,275		8,065	7	17
S	190	2,840	4,660		7,500	8	18
R	141	3,176	4,045		7,221	7	16
Q	92	3,146	3,430		6,575	6	15
P	42	2,900	2,720		5,620	6	15
O	- 7	2,880	2,260		5,140	5	15
N	- 56	673	2,260	700	3,633	5	16
M	- 105	1,430	2,260	1,310	5,000	5	17
L	- 154	2,020	2,260	1,930	6,210	7	19
K	- 203	2,140	2,260	2,550	6,950	7	14
J	- 253	1,790	2,260	3,150	7,200	8	15
I	- 302	2,090	2,260	3,765	8,115	9	17
H	- 351	1,946	2,260	4,380	8,586	9	18
G	- 400	1,748	2,260	4,840	8,488	10	20
F	- 450	1,684	2,260	5,616	9,560	10	20
E	- 499	1,684	2,260	6,230	10,174	11	22
Stripping							
AA	584	1,886	1,150		3,036	8	
Z	535	2,430	535		2,965	8	
Y	485	2,000	1,500		3,500	9	
X	436	2,520	1,570		4,090	10	
W	387	2,330	1,080		3,410	9	
V	338	2,300	1,280		3,580	9	
U	288	2,200	1,120		3,320	8	
T	239	2,790	1,480		4,270	10	
S	190	2,690	1,480		2,170	10	
R	141	2,260	1,480		3,740	9	
Q	92	2,220	1,655		3,875	9	
P	42	2,490	1,655		4,145	9	

Table 11. (Cont'd.)

Bench	Elevation Diff. ft	Distributed Hor. Dist. ft	Inclined Dist.(-) ft	Inclined Dist.(+) ft	Total Haulage Dist. ft	50-ton Trucks
O	- 7	2,430	2,000	-	4,430	10
N	- 56	1,900	2,000	700	4,600	11
M	-105	1,430	2,000	1,310	4,740	12
L	-154	2,020	1,200	1,930	5,150	12
K	-203	2,140	2,260	2,550	6,950	7
J	-253	1,790	1,200	3,150	6,140	7
I	-302	2,090	1,200	3,765	7,055	8
H	-351	1,946	2,260	4,380	8,586	9
G	-400	1,748	2,260	4,840	8,488	10
F	-450	1,684	2,260	5,616	9,560	10
E	-499	1,684	2,260	6,230	10,174	11

Production Hauling Costs

The production hauling costs are summarized in Table 12. Calculations are made by bench, and variations in cost depend principally on bench elevation, and in the size of the fleet.

Table 12. Summary of Hauling Operating Costs

Bench	Ore Tons	Waste Tons	Trucks in Stripping	Trucks in Ore production	Stripping Cost/ton	Ore Cost/ton	Overall Cost/ton
AA	1,790,000	14,420,000	8	10	$ 0.025	$ 0.070	$ 0.125
Z	2,320,000	14,420,000	8	9	0.025	0.063	0.118
Y	5,950,000	10,060,000	9	9	0.028	0.063	0.124
X	5,330,000	12,050,000	10	8	0.031	0.060	0.128
W	6,310,000	11,780,000	9	8	0.028	0.060	0.121
V	7,220,000	11,700,000	9	9	0.028	0.060	0.121
U	6,950,000	10,200,000	8	7	0.025	0.057	0.112
T	9,500,000	7,550,000	10	7	0.031	0.057	0.125
S	9,400,000	7,260,000	10	8	0.031	0.060	0.128
R	6,280,000	9,500,000	9	7	0.028	0.057	0.118
Q	6,720,000	7,960,000	9	6	0.028	0.054	0.115
P	5,150,000	8,640,000	9	6	0.028	0.054	0.115
O	4,050,000	8,230,000	10	5	0.031	0.051	0.119
N	1,288,000	9,450,000	11	5	0.034	0.051	0.126
M	4,620,000	4,480,000	12	5	0.037	0.051	0.132
L	6,690,000	1,120,000	12	7	0.037	0.057	0.138
K	4,470,000	2,240,000	7	7	0.022	0.057	0.105
J	5,310,000	1,260,000	7	8	0.022	0.060	0.108
I	1,112,000	1,910,000	8	9	0.025	0.063	0.118
H	1,146,000	1,146,000	9	9	0.028	0.063	0.124
G	974,000	810,000	10	10	0.031	0.070	0.138
F	0	1,398,000	10	10	0.031	0.070	0.138
E	580,000	466,000	11	11	0.034	0.073	0.148

From the above results it can be seen that the overall production costs are very uniform, oscillating between a minimum of $0.112/ton of ore, to a maximum of $0.148/ton

Average haulage cost/ton = $0.124/ton of ore.

APPENDIX D

CONCENTRATE TRANSPORTING TRUCKS

Ownership and operating cost per hour excluding depreciation (one truck) =	\$6,89/hr
Productivity (one truck) =	174 tpd

Ownership and operating costs per ton =

$$= \frac{\$6.89/\text{hr}}{7.25 \text{ ton/hr}} = \$0.95$$

Total cost per day = 523 ton x \$0.95/ton =	\$496.00

$$\text{Cost per ton-mile} = \frac{\$496.00}{523 \text{ ton} \times 74 \text{ miles}} = \$\ 0.013$$

Labor (9 drivers, 9 helpers, 4 extra men)

22 men x \$7.00/shift = \$154.00/day

$$\text{Cost/ton-mile} = \frac{\$154.00}{523 \text{ ton} \times 74 \text{ miles}} = \underline{\$0.004}$$

Total ownership and operating costs of trucks/ton-mile =	\$0.017

APPENDIX E

BLASTING AND DRILLING EQUIPMENT CALCULATIONS

Dimensions of block producing 30,000 tpd	85 x 85 x 49.2 ft
Area of block producing 111,500 tpd = = 111,500 ton/(4.16 ton/sq ft) =	26,800 ft^2
Dimensions of block =	164 x 164 x 49.2 ft

The drilling pattern used for calculations is shown in Fig. 41

Diameter of drill hole =	7.0 in.
Powder factor =	0.227 lb/ton
Volume of hole $= \dfrac{\pi \times (7/12)^2}{4} \times 55.6 =$	14.9 ft^3

Blasting agent	AN/FO
sp gr of AN/FO microprills	0.95
Number of drill holes	49

Daily powder requirement = 111,500 ton x 0.227 lb/ton =	12.65 ton
Weight of explosives = 62.4 x 0.95 x V x 49 =	25,300 lb
$V = \dfrac{25,300}{62.4 \times 0.95 \times 49} =$	8.62 ft^3
AN/FO column length = $8.62/(\pi D^2/4)$ = = 8.62/0.267 =	32.30 ft
Total required drilling length 49 d.h. x 55.6 ft/d.h. =	2,730 ft
Tons per ft of drill hole 111,500 ton/2,730 ft =	41

Drilling

The drilling rate is estimated @ 36 ft/hr. Operation of Lamco Co. in Liberia in a similar type of rock and conditions, drills at a variable rate of 19, to 54 ft/hr with an average rate of 36 ft/hr. This average is considered in these calculations. Data for Lamco Co. are given by Hartman (1969, p. 116).

Calculations:

Time for changing rig from hole to hole =	8 min
Time per hole = 56 ft/ft per hr = 56/36 = = 1.555 hr =	1 hr 33 min
Time allowed for meals in three shifts	1.5 hr
Preparation and loading time	2.0 "
Extra time losses in 24 hr	1.0 "
Total unproductive time	4.5 hr
Operating time	19.5 hr
Total time per hole = 1 hr + 33 min + 8 min =	1 hr 41 min
Holes per rig per day = 19.5 hr/ 1.684 hr per hole = 11.6 ≈	12
Number of rigs = 49 holes/12 holes per rig =	4
Extra units for maintenance purposes =	1
Total units	5

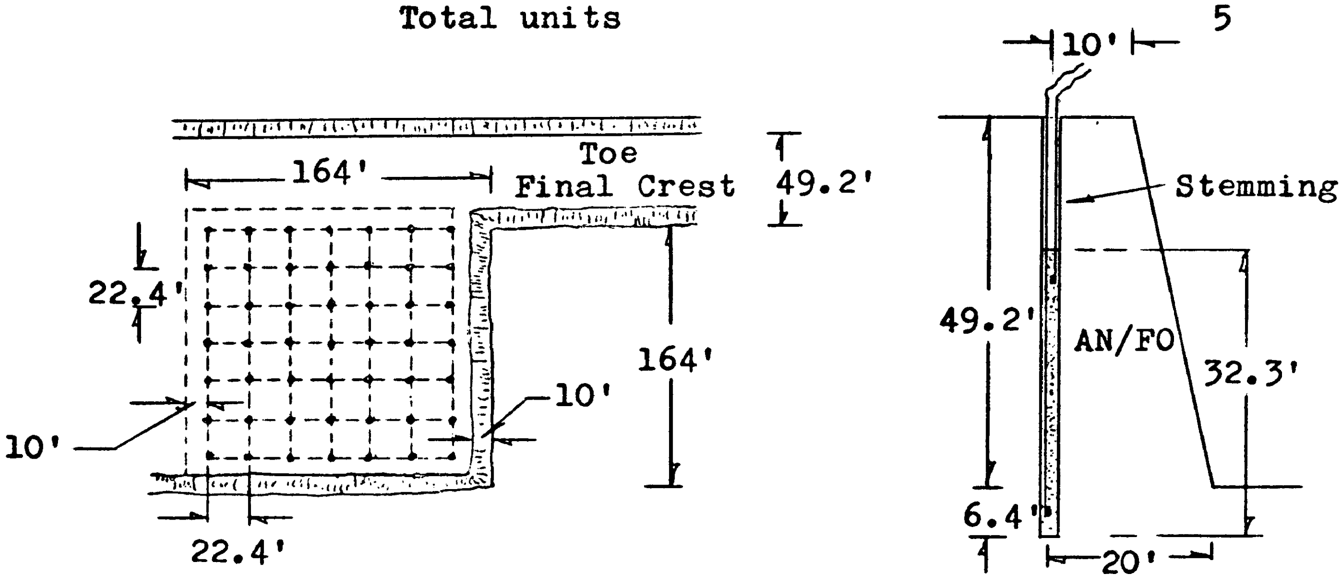


Fig. 41 Drilling Pattern

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