1210 0083

(48)
ITEM 83

Mack Forbes

Question: How much would it cost to set up and mine 10,000 tons of Leaching ore.

Process it, drill well 150 ft., build cement leach tanks (or others if a more economical way. To point of Reduction to cement copper.

Place Contact Nevada.

Roughly lay out steps.

Including your supervision.

Dec. 18, 1978.
Carson City, Nevada.

Frank Lewis.

A sample of ore marked 9-2-1 was crushed to ¼ inch, mixed and a Head Sample cut out for analysis.

A portion was Static Leached for 72 hours using a dilution ratio of one part of water containing 40 grams of Sulfuric Acid per liter to two parts of ore. After 72 hours the Pregnant Solution was decanted and the Leached Residue washed with water until the sample was neutral. A sample was cut from the Leached Residue for analysis. The Pregnant Solution was assayed for Copper, Iron and Silver.

An analysis of the Head Sample is listed below.

Total Copper	1.4%
Oxide Copper	1.3%
Iron	2.7%
CaO(Lime)	.60
SiO	62.50%
Silver	0.55 Oz/Ton.

From the analysis, this ore could be amenable to Heap Leaching with dilute Sulfuric Acid.

The following table sets forth the data obtained from Static Leachingwith dilute Sulfuric Acid.

Product	% Cu.	Lbs. of Cu. recovered per/ton of ore.	lbs. of Sulfuric Acid consumed per lb. of Copper recovered.
Pregnant Solution.	---	16.5	1.94
Leached Residue.	0.812	----	----
Calculated Heads.	1.636	----	----

% Extraction of Copper = 50.37.

This preliminary test looks encouraging. A test is now in progress using Static Leaching for the Copper followed by Heap Leaching for Silver.

Edward S. Shedd

10,000
25
$2500.00

175' ÷ 6 = 20.8
40 ÷ 6 = 6.6 } x 1320

Cost 1

Possible Costs.

b Bulldozer, over all 5 days 1500

Pre-leach

Surface samples plus assaying for oxide or acid soluable Cu
These may be composite { Total Cu / Au Ag } 50 = @ $10.00 500.00

Drill holes for rubblizing blast. as per Margrat's approximation 5000' @ $5.00/ft $5000.00
5134
includes sampling etc
~~Pre-leaching~~

192 Drill Holes to be sampled individually and composited
± 750 samples @ $10.00 7500.00

→ Powder cost of blast

→ Moving 10,000 Tons of Leach material to Leach pad @ $0.25/T 2500.00

proposed ~~leach material~~ to be

If the 10,000 Tons leached contains @ .05% soluable Cu there will be 100,000 lb Cu present and at
@ .60% Recovery = 60,000 lb Cu recovered which at
@ $0.60/lb = $36,000. recoverable Cu
70 - $42,000.00
75 - $45,000.00

Cementation launders wood or Concrete — 2,000 ?

Preparation of
Leach Pad of 1/2 acre
@ $11,000 per acre — $ 5,500

H_2SO_4 for ~~100,000~~ 60,000 lb
recoverable Cu @
~~3~~ lb acid / lb Cu
or 150 Tons Acid @ $30.00/Ton — 4,500

Iron ~~3~~ 1.5 lb Fe per lb of
~~recoverable~~ Cu cemented
or 10 Ton Fe @ $~~130~~ 135.00/Ton — 1,300

Solution make up
Water @ $0.10 per 1000
gallons @ 60 gal/minute
or $ 8.60/day or — 860
100 days @ —
Smelter and refining costs $0.30
0.29/lb cu produced
Generator or power supply — ?

Well 150' deep
water pump, ~~pipe~~ power &
storage tank, & distribution system — $ 5,000

1 or 2 acid proof pumps ±100 g/m — ? 5,000 ?

Smelter refining and selling
charges @ $.30/lb — $ 9000
for 30,000 lbs cu

Sub total — $ 45,160

Cat 3

Temporary

Sample lab set up
for wet coppers

chipmunk crusher pulverizer or bucking boards screens glass ware and reagents	garden houses ~~800~~	1173	
		1780	~~5000~~ ~~2000~~
		2953	3418
	Hot plate	~~240~~	4000.00
	Sample splitter	125	
	Balance	~~100~~	
		3 100.00	

Trailer for office
and lab used ——— 3000.00

Change room with ~~water~~ supply } 3000.00
+ First aid - specially for } 1600.00
possible acid burns

Pick up truck 3000.00

Staff -

Supervisor part time 10 ~~to 20~~ days ~~/month~~ $200/day -

assistant sampler lab
man etc - Recent graduate ± 1600.00 /month
or advanced student
capable of running
wet assays and ~~setting~~ up
and ~~running~~ sample met. tests on
the property.

Helper — days labor 1 or 2 — @ $32.00 /day
watch pads - help ~~with~~ sampler -
clean Precip tanks
dry ppt. & sack.

Misc.

~~Past Operations~~

The most important part of the whole program will be the correct determination of grade and % recovery as well as the maximum ~~fragment~~ size that ~~should be in the~~ ~~leach~~ heap material to be leached and amount of copper ~~we could be~~ recovered. Without this information it will be impossible to make a justifiable evaluation of the probable success of a leaching operation, either in place or in heaps.

Alternatives
Make several small pads and Crush to various sizes and for each — with potable crushing plant — No screening

1 Run of mine — 6"
2 — 4"
3 — 2"
4 — 1"

This will depend upon ~~met~~ continuing Met. testing

Drive a decline under rubbelized test area and collect leach solutions from in place (in situ) leaching

Introduction — Since final main thrust of ~~program~~ program appears to be drive is for possible in place leaching testing to find ~~[illegible]~~ the largest fragments possible

9½ lb / yard   40 x 30 x 100 ÷ 27

4500 yds

± 10,000 lb –

drill   6 x 6   3 – 3½" inch holes

6 x 6 = 36 x 30 –

40 yd per hole

100 holes – $100.00

$6500 – $7000.00

about $1.00

10,000 Tons
2000

2,000,000 lbs Cu
.05

10,000 0.00
.6

2000

60,000.0 lb Cu

60,000 lb Cu
3

180,000 lb Fe

90 Tons
2000)180,000
180.00

90 X 130 Ton 11,700

Recovery of Cu

559

1,650,000 ⟌ 30,000.00 = .02

1,650,000 ⟌ 10,000.000 = .006
990

⟌ 85,000.00 = .05

1650000 ⟌ 113,000.00 = .0
1155

.60 @ .05 % =

6.00 / Ton

@

60 % Recovery

6.00 x .60 = $3.60

60
360

1 Ton	recovers	@	100%	10 lb
1 "	"	"	60%	6 lb cu

$3.951 ÷ 6lb = $0.6585 / lb cu
or $0.659 / lb cu

Acid @ 3lb / lb cu
60,000 Cu x 3 = 180,000 lb H_2SO_4

2000 ⟌ 180,000 = 90 T/ton acid
18000

90 x $30 = 2700.00

60,000
.30
18,000.00

232,000.0

390,000.0

1,6500,000 | 6,523,000

1,6500,000 | 63,000.00 (.0)

1,650,000 | 165,000.0 (.10)
165000 0

1,650,000 | 285,000.0 (.17)

1,650,000 | 33,000.00 (.02)

1,650,000 | 112,000.0 (.07)

1,650,000 | 2,376,000.

1,650000 | 1,485,000.0 (.90)
1485000 0

1,650,000 | 63,000.00 (.04)

1,650,000 | 10,000.000 (.006)
990
100000

1,650,000 | 23,000.00 (.01)
19,000.000

19,000.000 (.00)

1,650,000 | 19,000.00 (.01)

165,000.0

285,000.0

728,000.0

48,000.00 (.03)

1,650,000 | 3376,000

D-1

See p 14 refining cost

Dump calculations

Tare Tracks

Taggart : ^swell of Broken rock in dumps or bins 95-100 lb / cu ft. or ^a swell of 1.6-1.9 times volume in place

Use 100 lb / cu ft.

1,6
1.9
3.5
1.75

12.5 cu ft / ton unbroken ore
@ 1.6 X = 20.00 cu ft / Ton
1.9 X = 23.75 cu ft / Ton

broken ore on pan 2 | 43.75
21.875

use 22 cu ft / ton for dump

which equals ± 90 lb per cu ft of dump.

p

@

Rock On - various 100-140 lb / cu ft.

@ 100 cu ft / Ton 1 T = 10' x 10' x 1'
@ 140 cu ft / " 1 T = 14' x 10' x 1'

10,000
100
1,000,000
50,000
20 | 1,000,000

@

100 cu ft / Ton · 10,000 T = 1,000,000 cu ft
for a 20' high heap

1 ft lift = 50,000 sq ft

Dimensions for 50,000 sq ft
could be ± 223.6' x 223.6'

heap base 225' x 225' for
vertical sides 20' high

For a frustrum of a pyramid
Volume = $\frac{1}{3}h\,(B + b + \sqrt{Bb})$

1

12.5 cu ft / Ton
10,000 Tons =
125,000.0 cu ft of unbroken ore

22 cu ft / Ton
10,000
220,000 cu ft / Ton

B-3

100000
100000
10000000000

$\frac{1}{3}h\left(B+b+\sqrt{Bb}\right)$

$\frac{20}{3}\left(62500 + 32400 + \sqrt{62500 \times 32400}\right.$

$\sqrt{2025000000}$

$\frac{20}{3} \times (94900 + 45000$

$\frac{20}{3} \times 139,900$

6.7×139900

$= 937,300.$ cu ft

$937,300 \div 22 = 42,605.9$ Tons

180/250

$\frac{X}{22} = 10,000$

$X = 220,000$

55225 □ft

235'

40000 □'

200

32400 □ft

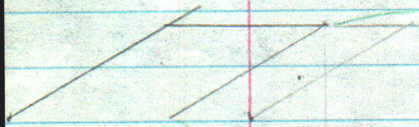
180

250

62,500 □ft

266

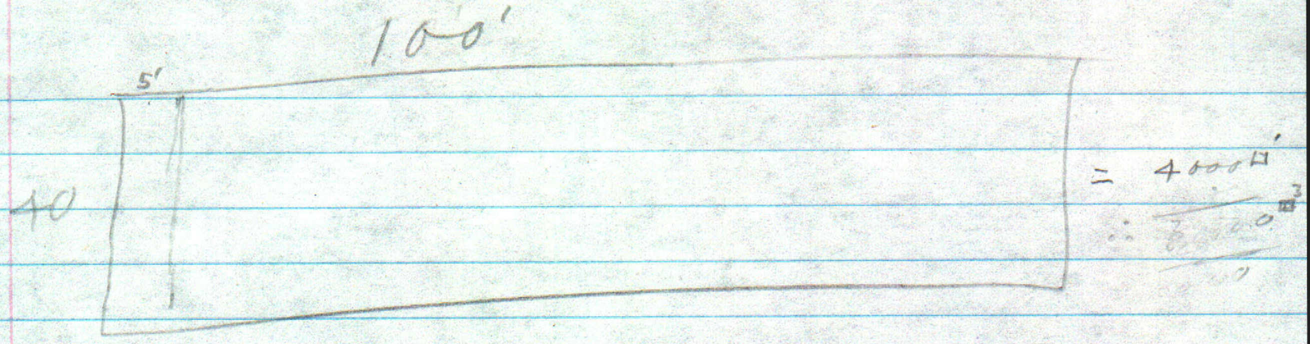
70756 □



300'

90,000 □ft

→ B-3



40 X 5 = 200 ÷ 12.5 = 16 T / Vert ~~ft~~

160 ÷ 5 = 20

16 X 20 = 320'

40' X 5' X 20' = 4000 ft³ = 320 Tons / vert ft

4000' X 31' = 124,000 ft³

1.6 X 124 000 = 198,400 ft□

1.9 X 124,000 235,600 ft□

434 000 ÷ 2 = 217,000 ft³

20/3 (55225 + 90,000 + √4970250000

20/3 (145,225 + 70500)

20/3 (215725) =

6.7 (215725 = 1,445,357.5

1,445,357.5 ÷ 22 = 65,698 Tons

Too much

10,000

135/300

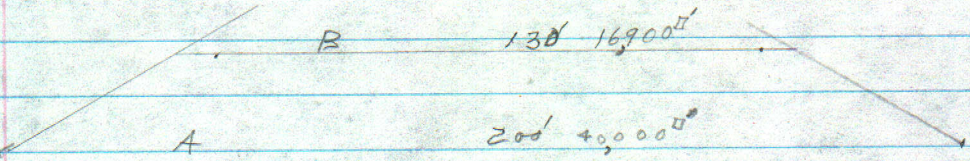
1.6
1.9
2) 3.5
1.75

ave 208.7'
on a side

D-5

124,000 ft^3 X 1.75 = swell 217,000 ft^3

start with 200,000 ft^3 and 20' lift



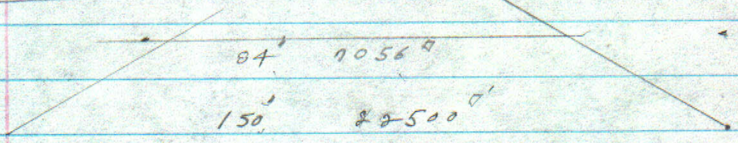
Too much

$\frac{20}{3}$ (16,900 + 40,000) + $\sqrt{AB}$ 676,000,000 =

8000
2000
64000000

6.7 X 56,900 + ~~26,000~~

6.7 X 82,900 = 555,430 ft^3 ÷ 22 = 25,245.8 T

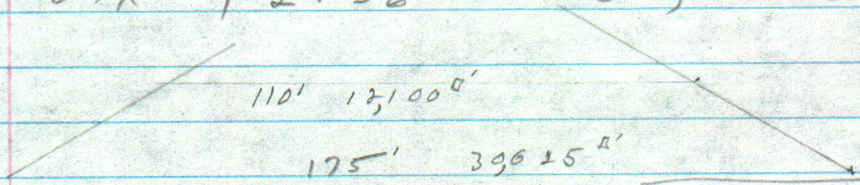


$\frac{20}{3}$ 7,056 + 22,500 + $\sqrt{AB}$ 158760000

6.7 X 29,556 + 12600

6.7 X 42156 = 282,445.2 ft^3 ÷ 22 12,838 T

1925

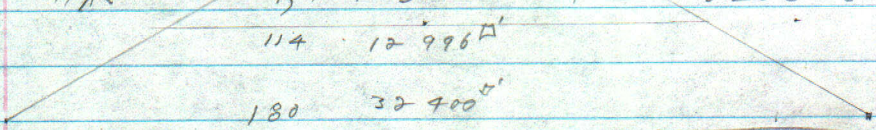


$\frac{20}{3}$ (12,100 + 30,625 + $\sqrt{AB}$ 370562500

6.7 X 42,725 + 19250

6.7 X 61,975 = 415232.5 ÷ 22 = 18,874 T

64888937



$\frac{20}{3}$ (12,996 + 32,400 + $\sqrt{AB}$ 421070400

6.7 X 45,396 + 20,520

6.7 X 65,916 = 441637 ÷ 22 = 20,074

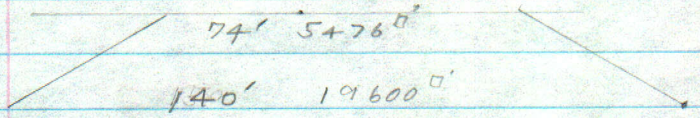
4 repose — ore — 38° – 42° stories

hard riprap – 45°

loose gravel + clay — 37°

in bins 100 – 140 lb / cu ft.

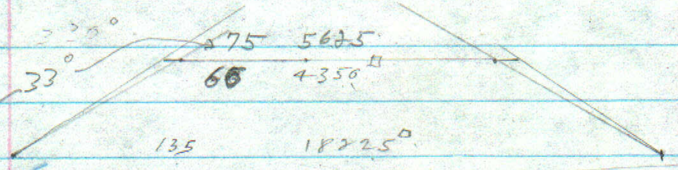
use 12 1/2 / 13 / cu ft.



20/3 19600 + 5476 + √AB 1,073,29,600

6.7 × (25076 + 10360)

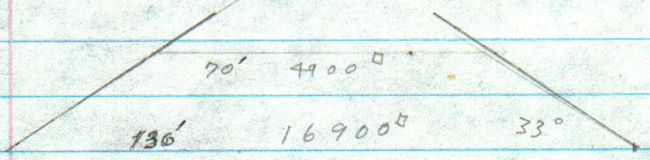
6.7 × 35436 = 237421.2 ÷ 22 = 10,792 T



30° 20/3 (4356 + 18225 + √AB 79,388,100 *

6.7 × (22581 + 8910)

6.7 × 31491 = 210989.7 ÷ 22 = 9,590. T



20/3 (4900 + 16900 + √AB 82810000

6.7 (21800 + 2878)

6.7 × 24678 = 165343 ÷ 22 = 7516 T

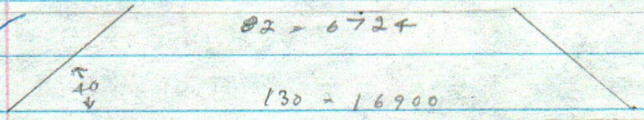
→ 33° (top 75′, base 135′)

3201.806

20/3 (5625 + 18225 + √AB 102,515,625 *

6.7 (23850 + 10125

6.7 × 33975 = 227632.5 ÷ 22 = 10,347 T



40° 20/3 (6724 + 16900) + √AB 113635600 .3370987900

6.7 (23624 + (+3371)

6.7 × 26995 = 180866.5 ÷ 22 = 8221 Tons

36411 ÷ 22 = 9,0433 T

6.7 67 + 169 + √11323

6.7 2364 1062.4 // 6.7 (342.4) = 229408 ÷ 22 10428 T

30°
66
135

$\frac{20}{3}(4356 + 18225) + \sqrt{79,388,100}$

6.7 (22581 + 8910)

6.7 X 31491 = 210989.7 ÷ 22 = 9590 T

33°
75'
135'

$\frac{20}{3}(5625 + 18,225) + \sqrt{102,515,625}$

6.7 (23850 + 10125)

6.7 X 33975 = 227632.5 ft^3 ÷ 22 = 10,347 T

Pad – Floor 135' X 135' 18225 □'

angle of repose 33°

height 20'

Pad Top 75' X 75' = 5625 □'

Volume 227633 ft^3

Tonnage factor for dumps 22 sqare feet per ton ∴

227633 ft^3 ÷ 22 ft^3 = 10,347 Tons

@
40°
82'
130'

Pad Floor 130 X 130 = 16900 □'

∡ repose 40°

height 20'

Pad Top 82 X 82 = 6724 □

Volume 180867 ft^3

8222 Tons

1 acre = 43,560 □' or 208.7' on a side

∴ a pad with ∡ of approx of 33° for ±10,000 T and 135' square at base (18225 □') 15' equal to $\frac{18,225}{43,560}$ = 0.42 of an acre.

135' [illegible]

43,560 □' ÷ 2 = 21780 □' = 148' on a side for ½ acre

Pad footing 150' square = + ½ acre.

in 1975 cost per acre for pads $11,000

TABLE 2. - Test blast summary

Quantity	Value
Number of blastholes	13
Blasthole diameter, in	9-7/8
Blasthole spacing, ft	14
Range of blasthole depths, ft	185 to 224
Average blasthole depth, ft	209
Average subdrilling, ft	7
Total drilling, ft	2,717
Explosive	ANFO
Average powder column depth, ft	202
Average stemming, ft	56
Average powder column length, ft	146
Average explosive/hole, lb	3,960
Average loading density, lb/ft	27.2
Total explosive, lb	51,500
Powder factor, lb/ton	2.2
Assumed overbreak, ft	4
Ore volume, yd^3	9,850
Ore weight, tons	19,700

Rubblizee a slot 100 X 40 X 30 deep

± 3" a 4"

WF

TABLE 2. - Test blast summary

Quantity	Value
Number of blastholes	13
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Johnson Camp - W.M Aug 1976 : p 48

0.8% Cu (0.50% acid soluble Chalcocite + chrysacolla,

leach 20' lifts 60-90 days

LIX plant 2,500 gal/min of feed 1.1 gm/liter Cu Ph 2.0-2.2

leaching sol. stored in plastic lined pons

pH leaching sol 8 to 9 gm/l H_2SO_4

now 6 gm/l

Drill Hole spacing 18' X 24' 6 3/4" bit.

Recovery LIX + Electro winning

Your figure of 1^{00} / Ton –
what does it include.

John Piscovich

Lewis

Assays

12-13-78 8:15 PM phone Assay on cut sample 9-2-1 - 1.40 total Cu
1.30 oxide Cu

@ ± 1/4" 2 day leach tails ~~assayed~~ ± 0.05 oz Ag
assayed 0.81 oxide Cu

Shed calculates 3 lb H_2SO_4 to recover 1 lb Cu or ± ~~$0.10 / lb~~ $0.0525
(For lb of Cu recovered -) acid at
$35.00 per ton. = $ 0.0175 / lb
$ 35. ÷ 2000 = $0.0175 then 3 lb Cu x $0.0175 = $0.0525
This sample (shed) low line

MEMO

Telephone Conversation with Hugh Clugle Jr. 707-987-3180

Conference - Those present were

Date 12-2-78 Time 8 A.M.

Place

Subject

will call tomorrow morning ((minute)

±$ ±0.28
102
0.29

#5 / Ton

$80.00 plus expen

916-635-8112

DU PONT

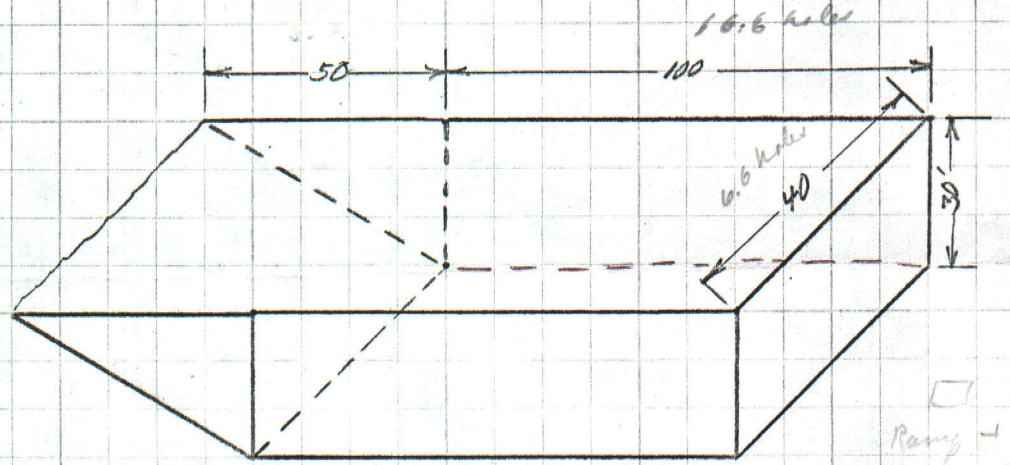
explosives

Location Elko, Nevada

Date 12/15/78

Representative

6h × 16h



Ramp + 10,000' + 2500 = 12,500 T

$27 \frac{ft^3}{yd} \div 12.5 \frac{ft^3}{T} = 2.16\ T$

Spacing - 6 ft
Burden 6 ft
Depth 30 ft - (Drill 33 ft)

$\frac{6 \times 6 \times 30}{27} = 40\ yd^3$

Tovex 800 - density 1.2 - Loads 3.7 lb/ft borehole

Load ± 27-28' - 104 lb/hole - ≅ 2.5 lb/yd³

$2.5 \div 2.16 = 1.16\ lb$ per Ton

Total # holes - 192 - 64 in ramp

$136 \times 33 = 4488$
$(56 \div 2)\,33 = 924$
5412'

136 hole @ 100 lb/hole ≅ 13,600 lbs
56 hole @ 40 lb/hole (average) ≅ 2240
15,840 lbs total

Timing: Shot opened at ramp - Timed to center.

E-08339

Printed in U.S.A.

50¢/lb.

STATE MINERAL PROFILES

Minerals in the Economy of Connecticut, by William A. Bonin and Hugo F. Thomas. 1979. 10 pp. 2 figs.

Minerals in the Economy of New Hampshire, by William A. Bonin and Glenn W. Stewart. 1979. 9 pp. 2 figs.

Minerals in the Economy of New York, by William R. Barton and Robert H. Fickies. 1979. 13 pp. 1 fig.

Minerals in the Economy of Utah, by William A. McKinney and Carlton H. Stowe. 1979. 17 pp. 2 figs.

PERIODIC REPORTS

Periodic reports dealing with various mineral commodities will be forwarded regularly if application stating in detail the need for certain reports is made to—

Branch of Editorial Services
Bureau of Mines
U.S. Department of the Interior
Washington, D.C. 20241

Some of these reports are issued weekly, monthly, quarterly, or annually. The following annual reports were issued during September.

Bromine in 1978 (Annual Advance Summary). 7 pp.

Iodine in 1978 (Annual Advance Summary). 5 pp.

Perlite in 1978 (Annual Advance Summary). 4 pp.

Potash in Crop Year 1979. 10 pp.

Stone in 1978 (Annual Advance Summary). 12 pp.

Pumice and Volcanic Cinder in 1978 (Annual Advance Summary). 6 pp.

III. Open File Report—NTIS

An open file report is an unpublished Bureau of Mines report that has been made available as reference material. Any open file report may be inspected during working hours at the locations indicated but may not be removed. If a number prefixed with PB is given, the report may be purchased from—

National Technical Information Service
U.S. Department of Commerce
Springfield, Va. 22161

Microfiche copies are $3.00 (domestic order) and $4.50 (foreign order). Paper copies are available at the prices indicated; double the price for a foreign order. Please order by numbers given.

OFR 82–79. Analysis of the Economic Feasibility for Development of Coal Resources in the Narragansett Basin of Rhode Island and Massachusetts, by Charles River Associates Inc. January 1979. 144 pp. 7 figs. This study assesses whether Naragansett Basin coal deposits could be developed commercially in the near future. To accomplish this, the potential market for the coal was identified, the price users would be willing to pay for Narragansett coal was assessed, and this price was compared with the expected costs of producing the coal. The study concluded that the major market, New England utilities, could burn bituminous coal from Pennsylvania or West Virginia at a lower total cost than Narragansett Basin coal, assuming expected mining conditions and emission policies. Therefore, the study concluded that Narragansett Basin coal resources could not be developed economically in the next 10 years. Research done under Contract No. J0188043 by Charles River Associates Inc. Available for reference at Bureau of Mines facilities in Denver, Colo., Twin Cities, Minn., Pittsburgh, Pa., and Spokane, Wash.; U.S. Department of Energy facilities in Carbondale, Ill., and Morgantown, W. Va.; National Mine Health and Safety Academy, Beckley, W. Va.; and National Library of Natural Resources, U.S. Department of the Interior, Washington, D.C. Order ONLY from NTIS: PB 298 165/AS; paper copy, $7.25.

IV. Journal Articles by Bureau Authors

The following articles have appeared in the outside press. Copies of these articles are NOT available.

OP 62–79. Tungsten Recovery From Searles Lake Brines by Ion Exchange, by P. B. Altringer, P. T. Brooks, R. O. Dannenberg, and W. N. Marchant. Min. Eng., August 1979, pp. 1220–1225. The Bureau of Mines conducted laboratory tests to devise a process for recovering a marketable grade of tungsten from the brine of Searles Lake, Calif. The brine contains only 56 ppm W (70 ppm WO_3). Using ion exchange resins synthesized by Bureau chemists, 98 pct of the tungsten was extracted from the brine and 92 pct of the tungsten was recovered as a marketable iron-tungsten concentrate containing 44 pct tungsten as tungsten trioxide. Although major technical problems have largely been resolved, research is progressing to escalate testing in a process research unit to provide more reliable data for an economic appraisal.

OP 63–79. Evaluating Ore Bodies for Leaching With Permeability Measurements, by Peter G. Chamberlain. Pres. at Ann. Meeting Soc. Min. Eng., AIME, New Orleans, La., Feb. 18–22, 1979, SME Preprint 79–27, 14 pp. Many mining engineers considering in-place leaching for the first time are uncertain as to how to evaluate ore bodies for potential leachability. Evidence presented in this report emphasizes the critical role of permeability in such evaluations. Whether an engineer is considering leaching "tight" formations typified by the porphry copper deposits of the southwest or the permeable uraniferous sandstones of Texas and Wyoming, permeability measurements are the keystone of any testing program. Permeability tests range from cheap and easy to highly expensive and sophisticated operations. Constant head, variable head, and pumping tests all have application in evaluating ore bodies for in-place leaching. Costs of these tests will not unduly deflate the pocketbooks of most mining companies. The basic procedures for selecting and running these field tests—including preparing suitable wells—are reviewed for the benefit of newcomers to in-place leaching.

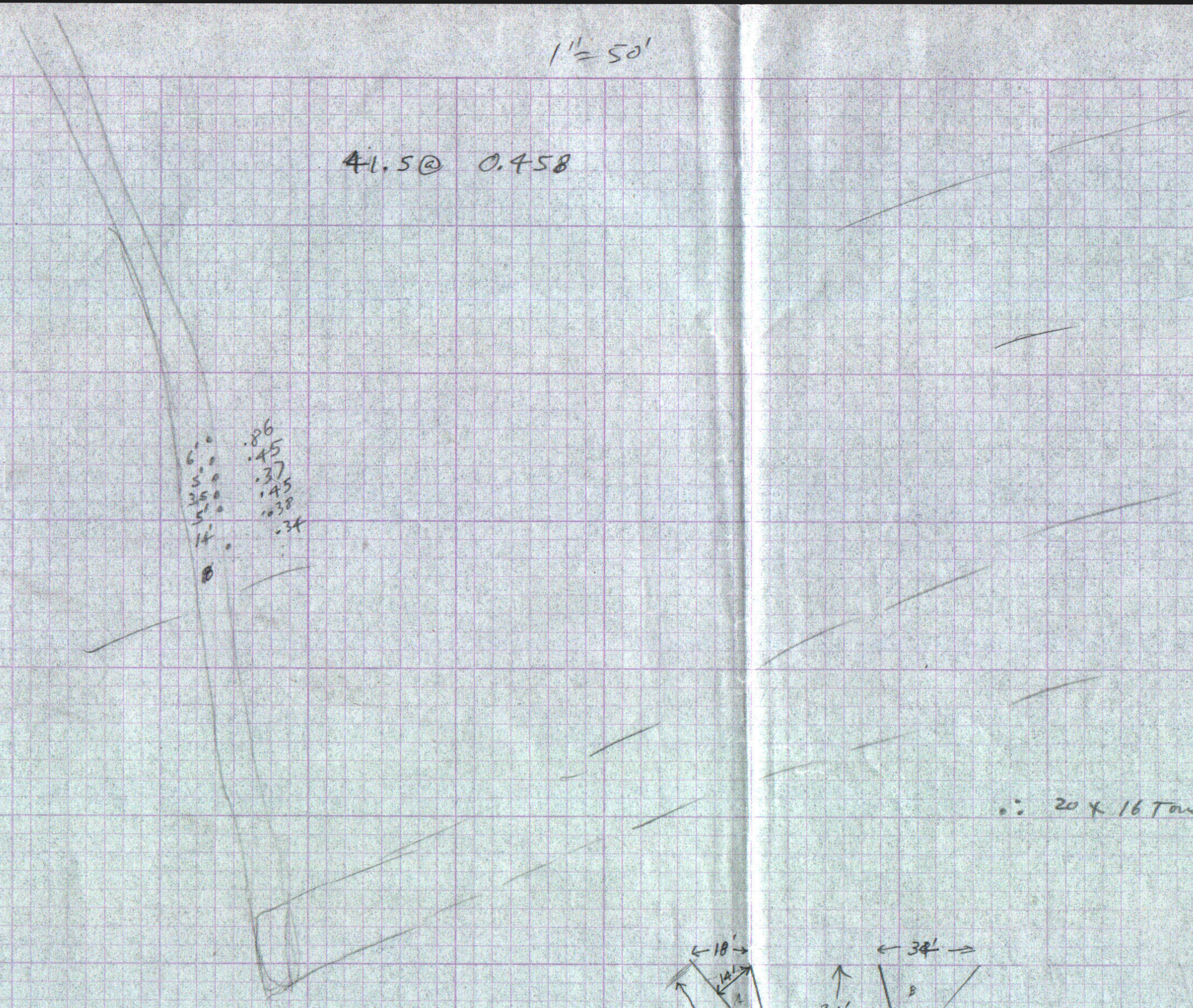
OP 64–79. Formation of Silicon Carbide From Silica Residues and Carbon, by Bing W. Jong. Am. Ceram. Soc. Bull., v. 58, No. 8, 1979, pp. 788–789. Recovery of alumina from abundant domestic alumina-bearing resources such as clay, anorthosite, alunite, shale, and dawsonite by leaching is currently undergoing miniplant testing by the Bureau of Mines. As a result of leaching the alumina minerals contained in these resources, a silica-rich residue becomes available.

OP 65–79. Borehole (Slurry) Mining of Coal and Uraniferous Sandstone, by George A. Savanick. Pres. at Ann. Meeting Soc. Min. Eng., AIME, New Orleans, La., Feb. 18–22, 1979, SME Preprint 79-53, 11 pp. The objective of this paper is to review advances in the art of borehole (slurry) mining made by the Bureau of Mines. Historical and general background information on borehole mining is given. The borehole mining concept is defined and justified in terms of economics, health and safety, the environment, and resource conservation. This is followed by a description of the design of a prototype borehole mining tool (BMT) developed by the Bureau of Mines. Next, the application of the BMT in the extraction of metallurgical coal from steeply pitching seams near Wilkeson, Wash., during 1975–76 is described. Finally, the Bureau efforts in applying borehole mining to the extraction of uraniferous sandstone is described. BMT design changes appropriate to uranium mining are described along with production, reliability, and reclamation data taken during 1977–78.

OP 66–79. Geochemical Changes During In Situ Uranium Leaching With Acid, by Daryl R. Tweeton, Gregory R. Anderson, Jon K. Ahlness, Orin M. Peterson, and William H. Engelmann. Pres. at Ann. Meeting Soc. Min. Eng., AIME, New Orleans, La., Feb. 18–22, 1979, SME Preprint 79-43, 27 pp. The Bureau of Mines measured the geochemical changes as H_2SO_4 was used for in situ uranium leaching by Rocky Mountain Energy Co. near Casper, Wyo. Cores and ground water were analyzed before leaching. Water samples were taken from observation wells located between injection and production wells as the leach solution was brought up to full strength in several steps. Measurements were made of pH, Eh, temperature, conductivity, total dissolved solids, dissolved oxygen, HCO_3, U, V, Na, K, Ca, Mg, SO_4, Cl, Mo, Mn, Fe, Al, Si, F, P, As, and Se. The data were gathered to assist in geochemical modeling of leaching and to study the potential environmental effects of acid leaching. Environmental considerations appear favorable. For example, the concentration of Se, a toxic element found in uranium deposits, stayed below the Environmental Protection Agency standard for drinking water.

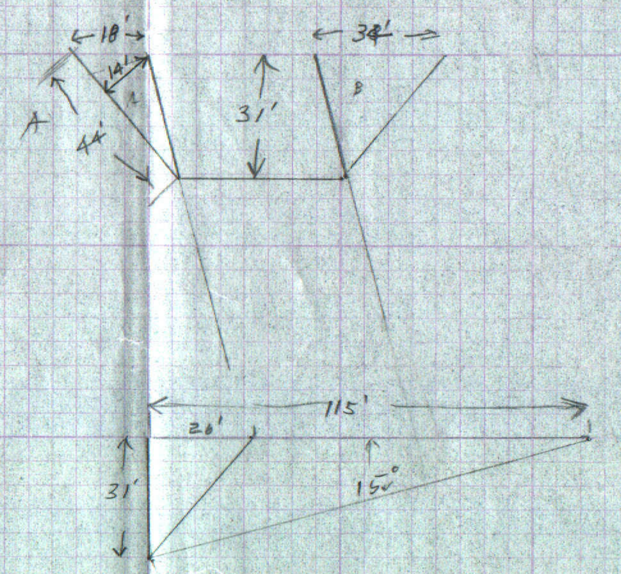
1" = 50'

41.5@ 0.458



40' X

a slot 40' long x 5' wide = 200 ft²
1 ft deep @ 12½ ft²/Ton = 16 Tons/vert ft
(200' ÷ 12.5)
100' long ÷ 5' wide = 20 slots ∴ 20 x 16 tons
∴ 20 x 16 Tons for 100' long = 320'/Vert ft
10' deep = 3200 Ton
20' deep = 6400 Ton
30' deep = 9600 Ton
± 31' deep = 10,000 Tons



A = $\frac{44}{2}$ x 14 = 308 ft² ÷ 12.5 = 24.6 T/long. ft
100' long = 2400 Tons

B = $\frac{34}{2}$ x 31 = 527 ft² ÷ 12.5 = 42.2 T/long. ft
100' long = 4220 Tons waste

40
10

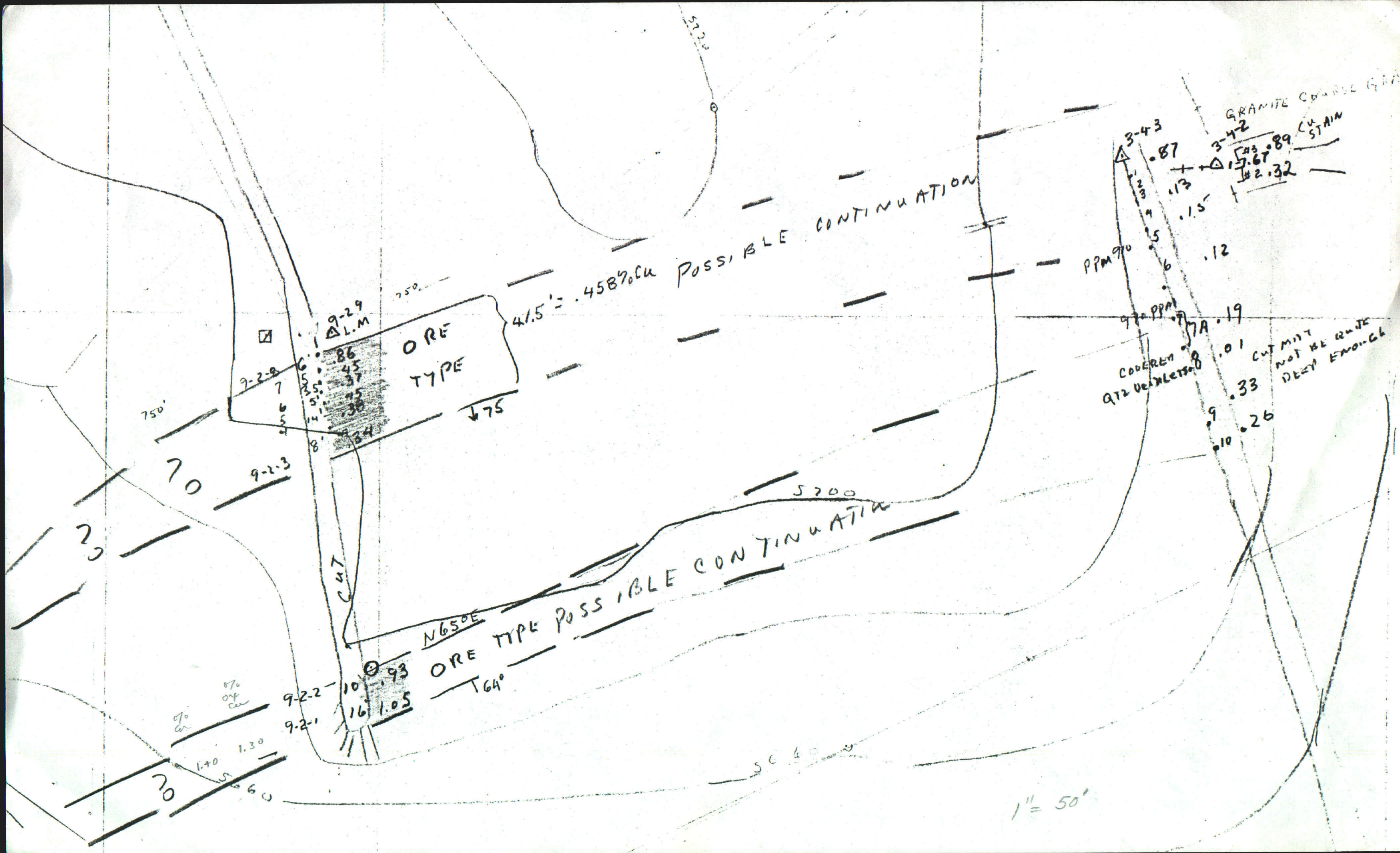
End C
End C'

$\frac{31}{2}$ x 26 = 403 ft²/long. ft. ÷ 12.5 = 32 T/long. ft X 40 = 1280 Tons waste

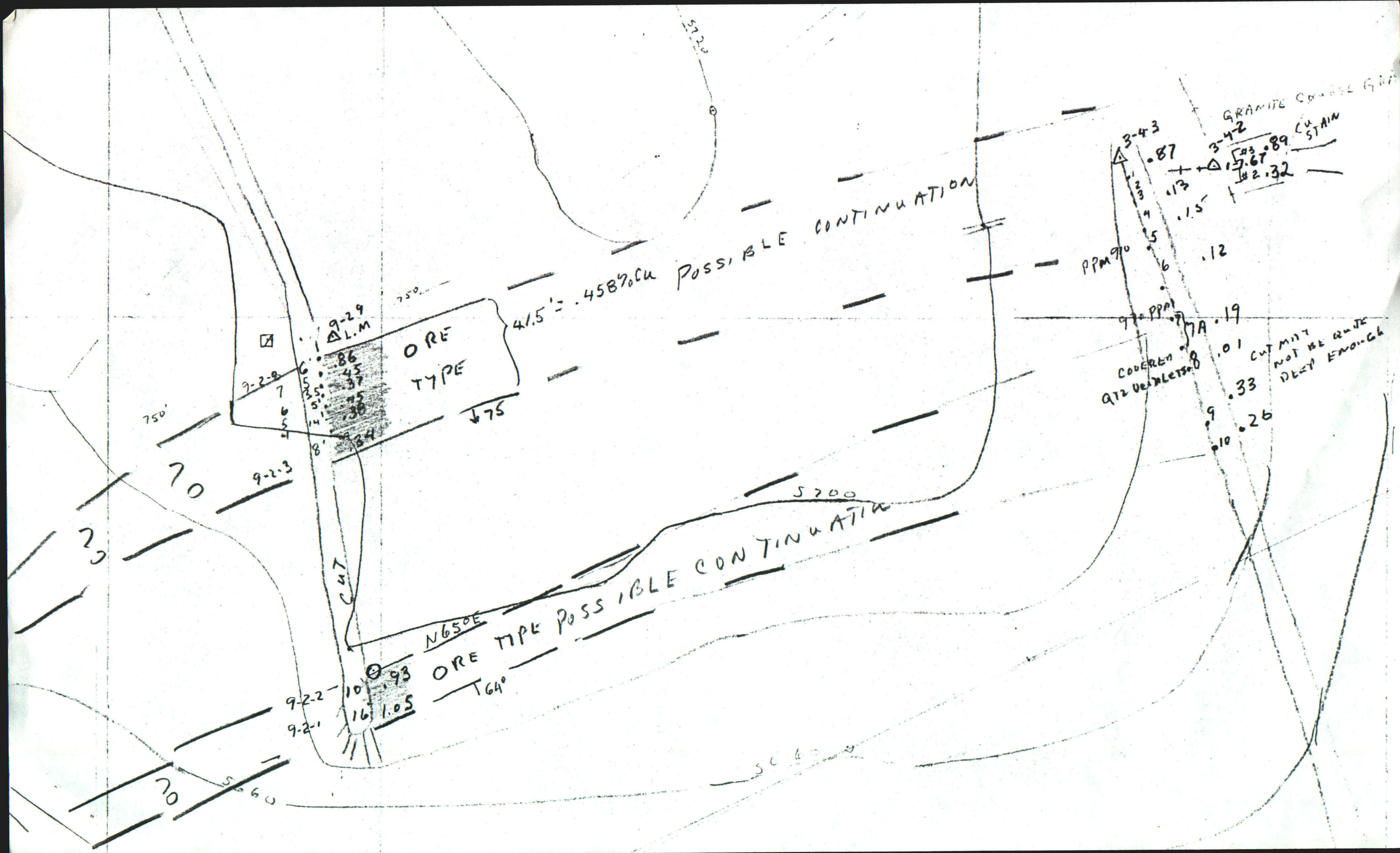
$\frac{31}{2}$ X 115 = 1783 ft²/long. ft ÷ 12.5 = 143 T/long. ft X 10 = 1430 Tons waste

waste Ends C + C' = 6930 Tons

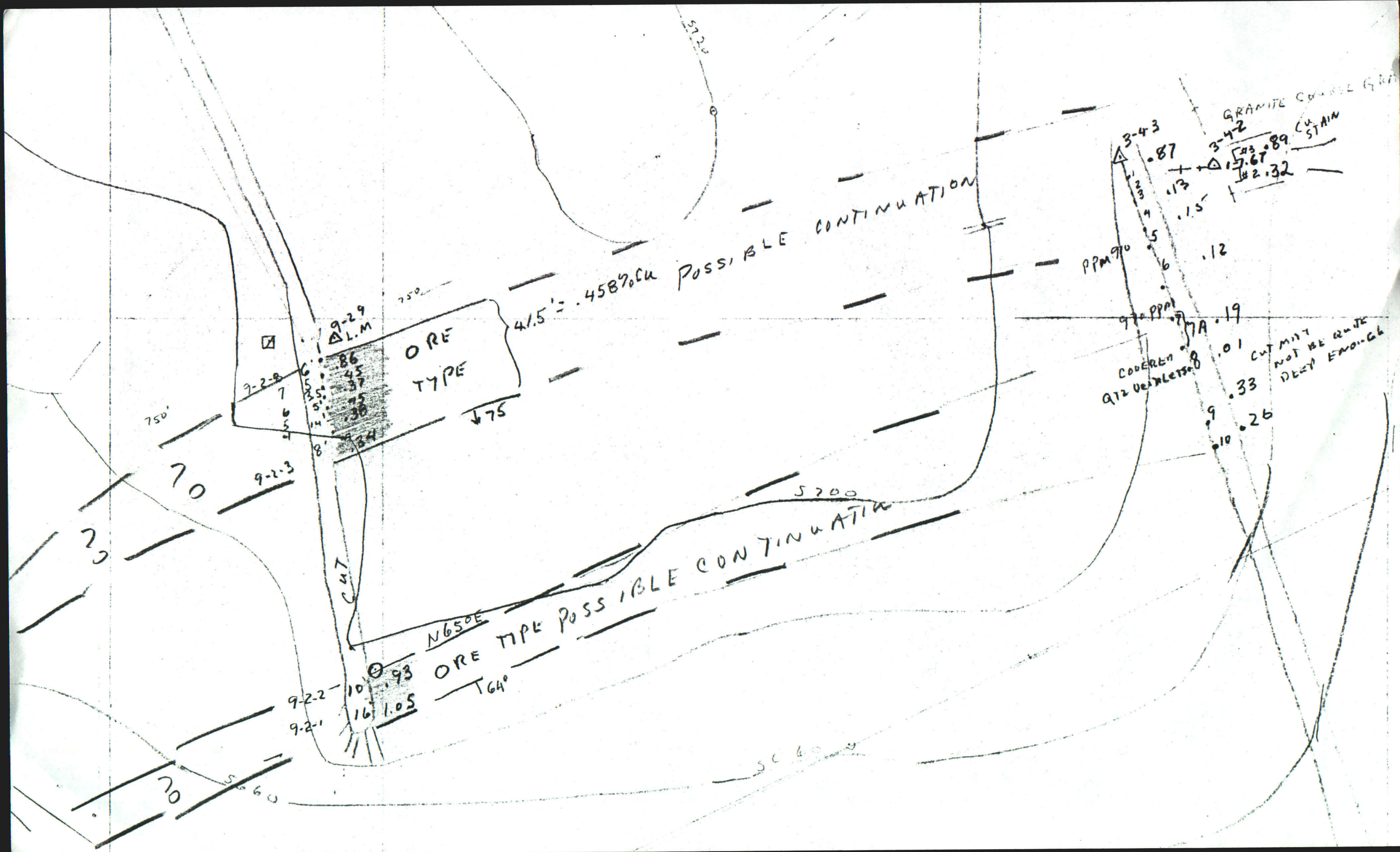
Franklin Hoar



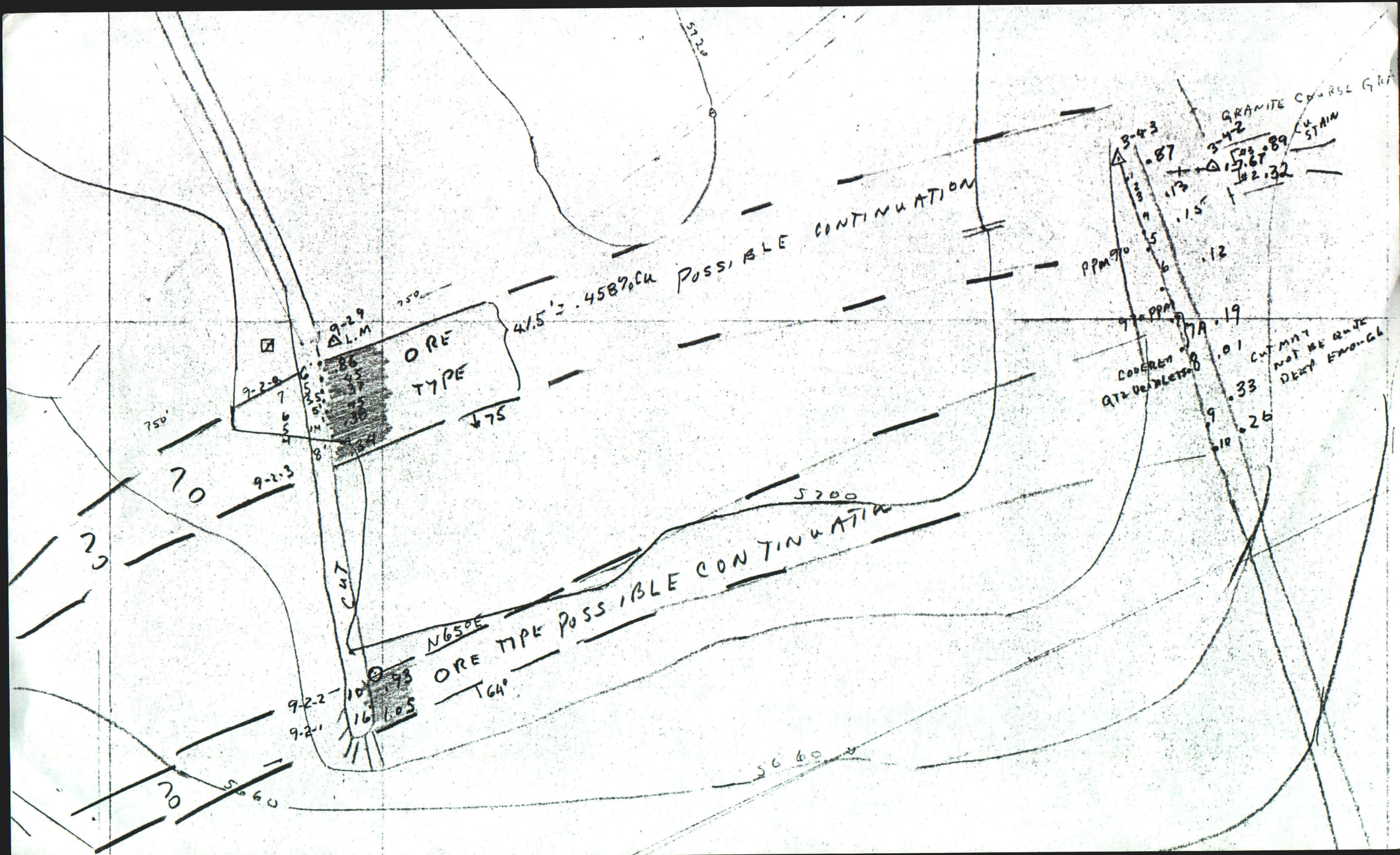
ORE TYPE
41.5' = .458% Cu POSSIBLE CONTINUATION
9-29
L.M
9-2-8
9-2-3
↓75
750'
70
ORE TYPE POSSIBLE CONTINUATION
N65°E
S70°
CUT
9-2-2
9-2-1
10'
16'
.93
1.05
T64°
1.40
1.30
3-4-3
3-4-2
.87
.13
.15
.12
.19
.01
.33
.26
.89
2.32
Cu STAIN
GRANITE
PPM 90
COVERED
QTZ VEINLETS
CUT MAY NOT BE QUITE DEEP ENOUGH
1" = 50'



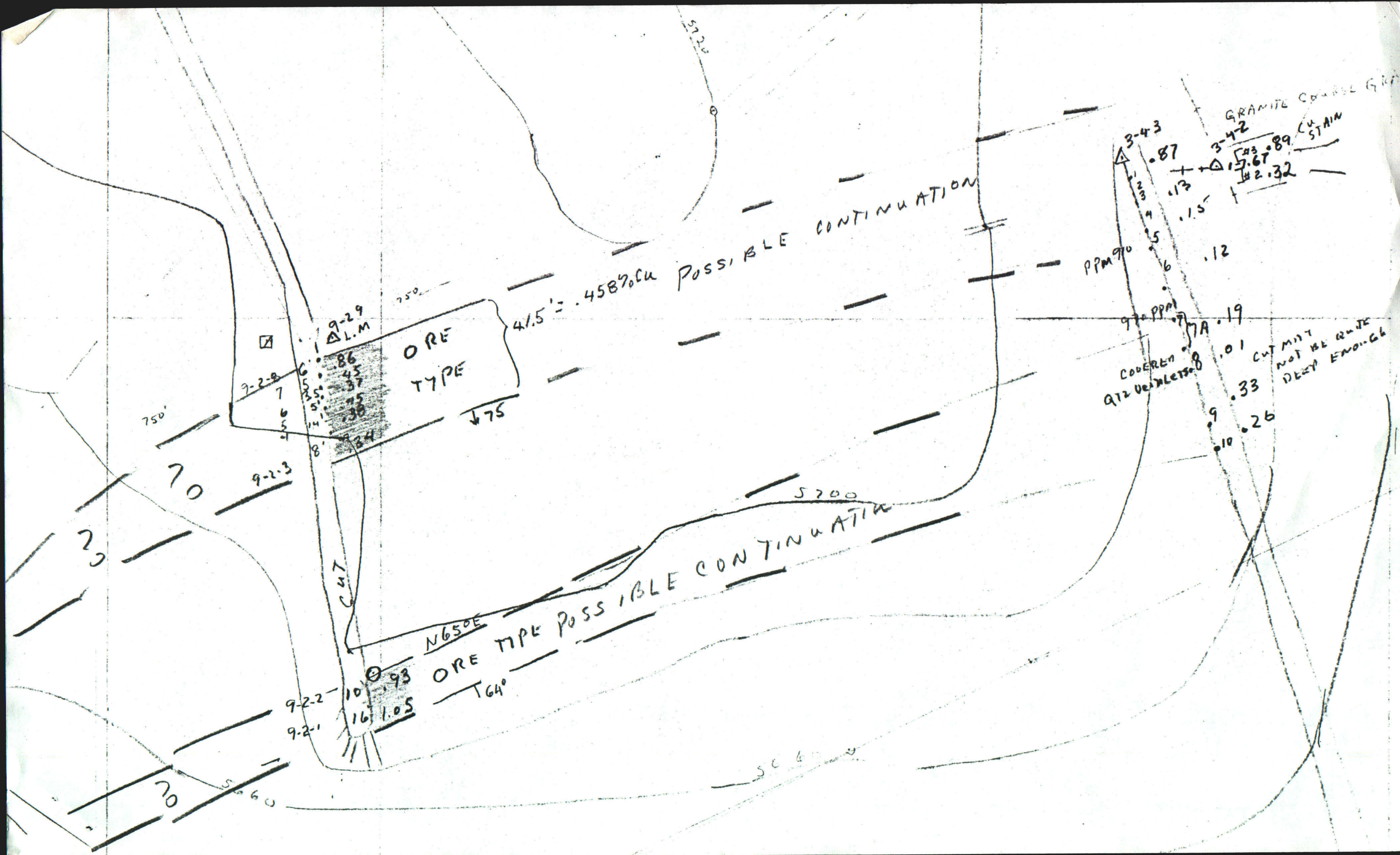
9-2-9
L.M
ORE
TYPE
41.5' = .458% Cu POSSIBLE CONTINUATION
9-2-8
9-2-3
.86
.43
.37
.45
.38
.34
↓75
750'
CUT
N65°E
S200
ORE TYPE POSSIBLE CONTINUATION
.93
1.05
9-2-2
9-2-1
T64°
3-4-3
.87
.13
.15
.12
.19
.01
.33
.26
3-4-2
#2 .32
.89
Cu STAIN
GRANITE
PPM
COVERED
QTZ VEINLETS
CUT MAY NOT BE DEEP ENOUGH



9-2-9
L.M
ORE
TYPE
41.5' = .458% Cu POSSIBLE CONTINUATION
75
750'
9-2-8
9-2-3
9-2-2
9-2-1
.86
.45
.37
.45
.38
.34
CUT
N65°E
.93
1.05
ORE TYPE POSSIBLE CONTINUATION
64°
S700
3-4-3
.87
.13
.15
.12
.19
.01
.33
.26
PPM Au
COVERED
QTZ VEINLETS
CUT MAY NOT BE QUITE DEEP ENOUGH
3-4-2
.89
.67
#2.32
Cu STAIN
GRANITE



4/.5' = .458% Cu POSSIBLE CONTINUATION
ORE TYPE
ORE TYPE POSSIBLE CONTINUATION
N65°E
CUT
9-2-9
L.M
9-2-8
9-2-3
9-2-2
9-2-1
3-4-3
3-4-2
GRANITE COURSE GR
Cu STAIN
CUT MAY NOT BE QUITE DEEP ENOUGH
QTZ VEIN LETS
5720
5700
5660
750'



ORE TYPE
41.5' = .458% Cu POSSIBLE CONTINUATION
ORE TYPE POSSIBLE CONTINUATION
CUT
N65°E
S200
9-2-9
L.M
9-2-8
9-2-3
9-2-2
9-2-1
.86
.45
.37
.75
.38
.34
.93
1.05
750'
3-4-3
3-4-2
.87
.13
.15
.12
.19
.01
.33
.26
.89
#2.32
PPM 90
Cu STAIN
GRANITE
CUT MAY NOT BE QUITE DEEP ENOUGH

Separate eggs

Pieces &

eggs –

Do not

screen

Relatively large

test pads to

get some heating

= temperature test below

Sprinklers
tubes ~
powder

also tablets
4 oo g / m

+ 8 % evaporation

P 3

1 6 g / m / sq ft

1 5 / m / sq ft

No attempt
made to go
into chemistry

Rock + gangue
Ls or ???

Pyrite for
acid

assumed must do
work by lease
or contract or
~~at least heavy
equipment leased~~
for 4 month period

Reclaim under
leaching area

Call Rice

Conco

50
32
18

5

21, 21, 21, 21,
21, 21, 21, 21,
21, 21, 21, 21,
21, 21, 21, 21, 21