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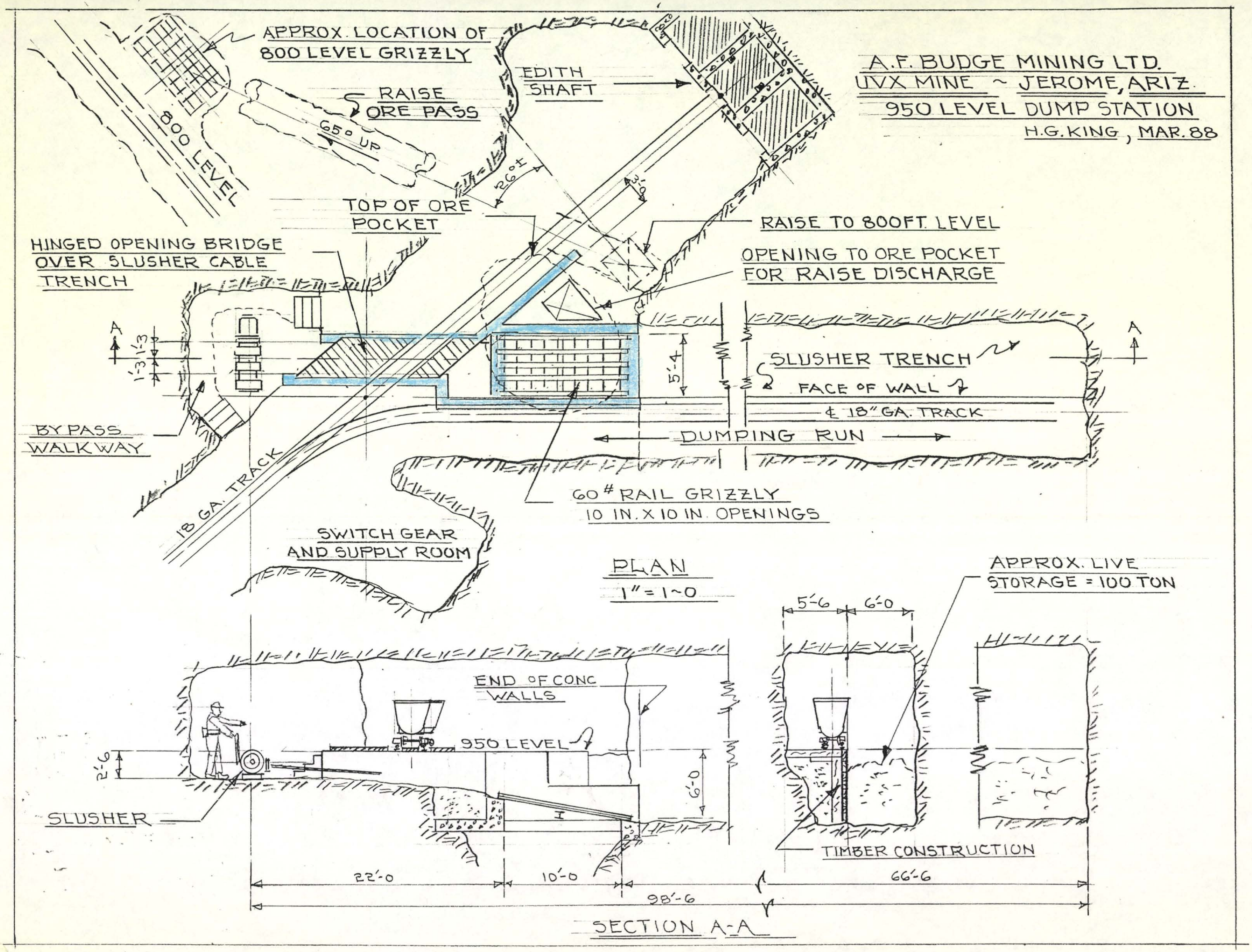


A.F. BUDGE MINING LTD.
UVX MINE ~ JEROME, ARIZ.
950 LEVEL DUMP STATION
H.G. KING, MAR. 88
APPROX. LOCATION OF
800 LEVEL GRIZZLY
800 LEVEL
RAISE
ORE PASS
65° UP
EDITH
SHAFT
26°±
3'-0
TOP OF ORE
POCKET
RAISE TO 800FT. LEVEL
OPENING TO ORE POCKET
FOR RAISE DISCHARGE
HINGED OPENING BRIDGE
OVER SLUSHER CABLE
TRENCH
A
1'-3
1'-3
5'-4
SLUSHER TRENCH
FACE OF WALL
℄ 18" GA. TRACK
DUMPING RUN
BY PASS
WALKWAY
18 GA. TRACK
60# RAIL GRIZZLY
10 IN. X 10 IN. OPENINGS
SWITCH GEAR
AND SUPPLY ROOM
PLAN
1" = 1~0
APPROX. LIVE
STORAGE = 100 TON
5'-6
6'-0
END OF CONC
WALLS
950 LEVEL
2'-6
6'-0
SLUSHER
TIMBER CONSTRUCTION
22'-0
10'-0
66'-6
98'-6
SECTION A-A

Copy BFD

MEMORANDUM

To: Don White
From: Paul Lindberg ← "Expert" visitor - geol. w/25-30 yrs exp. Has done much work near Jerome
Date: February 1, 1986
Subject: MAPPING OF A PORTION OF THE 1100 LEVEL, U.V.X. MINE, JEROME, ARIZONA

On January 24, 1986 I made a very brief geologic examination of a portion of the U.V.X. 1100 foot Level. Time did not permit any more than a cursory look at a small part of the recently opened up level. In the company of Glenn Davis of CoCa Mines Inc. we made the traverse shown on the accompanying sketch map.

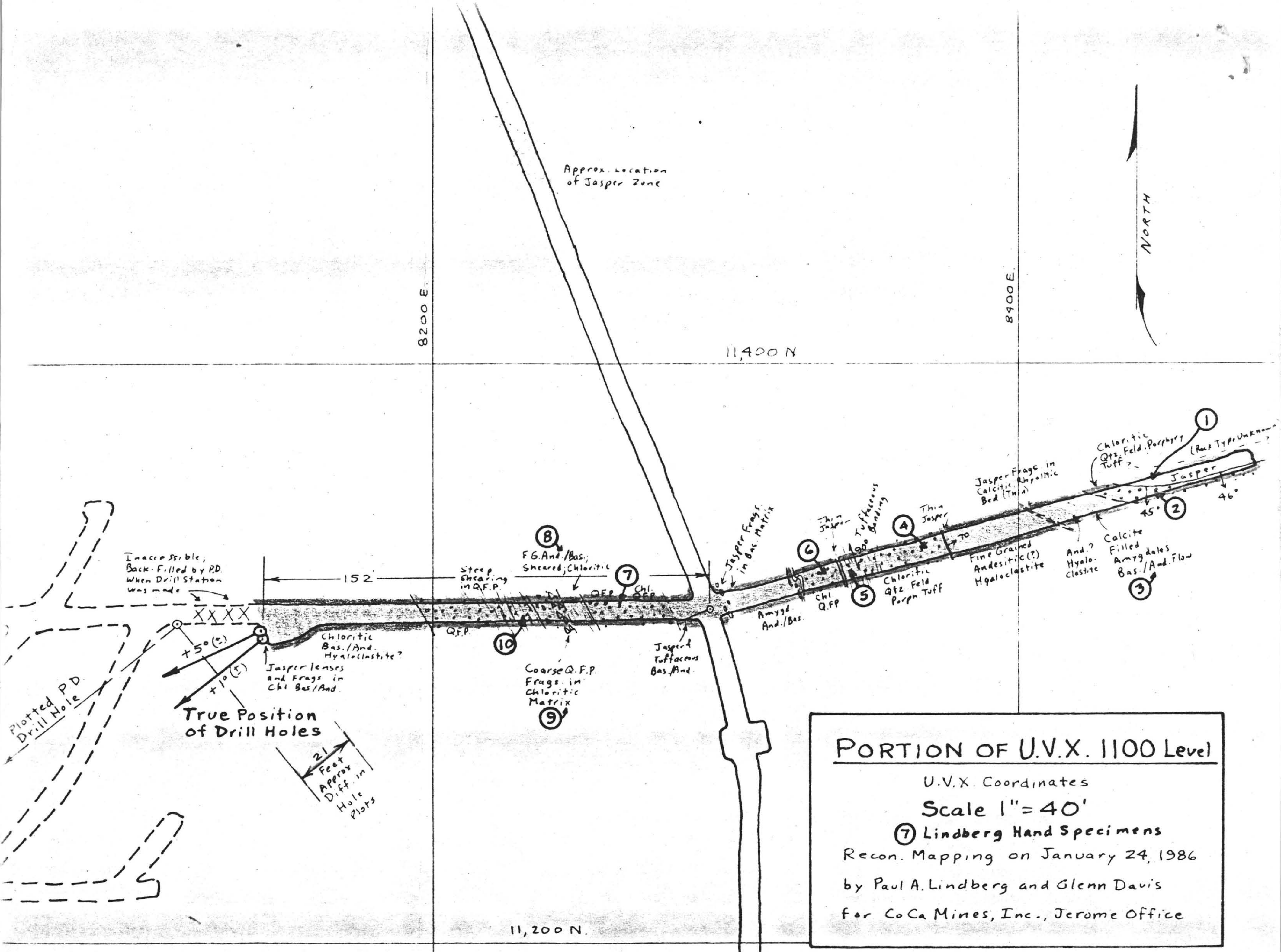
Ten rock samples were collected during the traverse; numbered 1 to 10. Nothing has yet been done with these hand specimens, but it is my intent to saw each one, spray lacquer its surface, cut blanks for possible future thin section use, and mount the pieces on a rock board which accompanies a sketch map of their location.

It was originally thought that in the brief period of time that we had available we would simply examine the rocks of the 1100 Level in a cursory way without intent to do any detailed mapping. However, the thought occurred to me underground that a useful approach might be for us to examine the drift out to the east of the Phelps Dodge drill holes so that the drill core data (drilled to the west) would augment our studies off to the east as far as the drift carries. We were thus able to "extend" the drill data another 343 feet to the east of the hole collars. During the measuring-in procedure it was discovered that the collar of the Phelps Dodge drill holes were plotted 30 feet too far to the ~~east~~ west from their actual location. In addition, the collars are 3 and 5 feet south of the southern edge of the drift and not originating at the edge of the drift as was shown on the Phelps Dodge maps. This means that in the direction of drilling the P.D. hole plots would have been located approximately 21 feet too far with respect to mine stopes that were encountered.

The rocks we saw included massive jasper, chloritic quartz feldspar porphyry, fragmental quartz feldspar porphyry, amygdaloidal andesite/basalt, andesite/basalt hyaloclastite, and associated hyaloclastites which contain jasper fragments. Some relatively thin, partially discontinuous jasper beds were also seen. It is highly probable that the short traverse passes through one or more fold closures, since some of the rocks encountered along the drift look identical to units further along. The thick and massive jasper seen at the extreme eastern end of the drift may or may not be the same as the one further to the north toward the Audrey shaft. Thickening and thinning of the jasper beds and fold complexities probably makethis connection nebulous based on the existing map data.

The dip of the massive jasper bed is to the S.S.E. at 45-46° and is positive evidence that cross folding is taking place. Correlation work on the 1300 Level U.V.X. data shows a distinct cross syncline passing just to the south of this point, and the dip reversal should come as no surprise. Overall the U.V.X. ores and rock units will plunge to the N.N.W., but local reversals such as this are to be expected.

Paul A. Lindberg



Approx. Location of Jasper Zone
NORTH
8200 E.
8900 E.
11,400 N
11,200 N.
Inaccessible; Back-Filled by P.D. When Drill Station was made
152
Steep Shearing in Q.F.P.
8
F.G. And./Bas.; Sheared; Chloritic
7
QFP
Chl. QFP
Q.F.P.
Chloritic Bas./And. Hyaloclastite?
10
Coarse Q.F.P. Frags. in Chloritic Matrix
9
Jasper lenses and Frags in Chl Bas/And
+5°(±)
+1°(±)
Plotted P.D. Drill Hole
True Position of Drill Holes
21 Feet Approx Diff. in Hole Plots
Jasper & Tuffaceous Bas./And.
Jasper Frags in Bas. Matrix
Amygd. And./Bas.
6
Chl Q.FP
5
Chloritic Qtz Feld Porph Tuff
Thin Jasper
Tuffaceous Banding
4
Thin Jasper
TO
Fine Grained Andesitic(?) Hyaloclastite
Jasper Frags in Calcitic Rhyolitic Bed (Thin)
And.? Hyalo. Clastite
Calcite Filled Amygdales Bas./And. Flow
3
Chloritic Qtz. Feld. Porphyry Tuff
1
(Rock Type Unknown)
Jasper
45°
2
46°
PORTION OF U.V.X. 1100 Level
U.V.X. Coordinates
Scale 1" = 40'
7 Lindberg Hand Specimens
Recon. Mapping on January 24, 1986
by Paul A. Lindberg and Glenn Davis
for CoCa Mines, Inc., Jerome Office

Worst Case Scenario

Assumptions and Parameters

gold	$475.00	per ounce
silver	$6.50	per ounce
M-3 & 902 Area Reserves	35000	tons
grade (gold)	0.15	oz/t
grade (silver)	3.7	oz/t
809 Area Reserves	35000	tons
grade (gold)	0.20	oz/t
grade (silver)	1.9	oz/t
Total Potential Reserves	70000	tons
grade (gold)	0.18	oz/t
grade (silver)	2.80	oz/t
cost, mining rock	$40.00	per ton
Recovery in CIL (gold)	0.9	90%
Recovery in CIL (silver)	0.75	75%
Processing, CIL	$9.00	per ton
Crushing	$2.50	per ton

UVX Mine Options (January 22, 1988)

	M-3 & 902 Areas	809 Area	Sub-total M-3, 902 & 809 Areas
Gross Revenues	$2,875,688	$3,316,688	$6,192,375
Operating Costs: -			
Mining @ $40/ton	$1,400,000	$1,400,000	$2,800,000
Crushing at $2.50/ton	$87,500	$87,500	$175,000
Processing @ $9/ton	$315,000	$315,000	$630,000
Operating Profit	$1,073,188	$1,514,188	$2,587,375

Assumptions and Parameters

gold	$450.00	per ounce	
M-3 Area High Grade	56400	tons	D.C.W. & R.W.H.
grade in Au equivalent	0.33	oz/t	July 3, 1987
iron content	15	%	
M-3 Area Low Grade	58400	tons	D.C.W. & R.W.H.
grade in Au equivalent	0.10	oz/t	July 3, 1987
iron content	18	%	
Total M-3 Reserves	114800	tons	
grade in Au equivalent	0.21	oz/t	D.C.W. & R.W.H.
iron content	16	%	July 3, 1987
809 Area Reserves	41000	tons	
grade in Au equivalent	0.26	oz/t	D.C.W. & R.W.H.
iron content	9	%	September, 1987
Total Reserves	234400	tons	9-02-87 Memo
grade (gold)	0.24	oz/t	C.A.O.
grade (silver)	2.88	oz/t	
grade in Au equivalent	0.29	oz/t	
Total Reserves	468000	tons	September, 1987
grade in Au equivalent	0.22	oz/t	projections D.C.W. & R.W.H.
Remaining Reserves (CAO)	146000	tons	
grade in Au equivalent	0.29	oz/t	
Remaining Reserves (DCW&RWH) without M-3 low grade	312200	tons	
grade in Au equivalent	0.22	oz/t	
cost, mining rock	$60.00	per ton	
Gold Recovery in Plant	0.85	85%	
Processing, CIL	$11.50	per ton	

High grade [illegible]

468,000
− 114

UVX Mine Options (September 3, 1987)

Ore to CIL Plant at UVX	M-3 Area High Grade	809 Area	Remaining Reserves CAO	Remaining Reserves DCW&RWH
Gross Revenues	$7,119,090	$4,077,450	$16,195,050	$26,271,630
Capital	$1,500,000			
Operating Costs: -				
Mining @ $60/ton	$3,384,000	$2,460,000	$8,760,000	$18,732,000
Processing @ $11.50/ton	$648,600	$471,500	$1,679,000	$3,590,300
Operating Profit	$3,086,490	$1,145,950	$5,756,050	$3,949,330
Recovery of Capital	($1,500,000)			
Sunk Costs (to 7-31-87)	($1,860,000)			
Additonal Exploration Development and Drilling	$0	($200,000)	($900,000)	($900,000)
Net Profit on Project	($273,510)	$945,950	$4,856,050	$3,049,330
Cumulative Profit	($273,510)	$672,440	$5,528,490	$3,721,770

27,391,590.

12,604

234,900 x 60 = 14,094,000 ?

Operating Profit

Gold Price per ounce	Mining Cost per ton	Gold Recovery percent	Processing Cost per ton	M-3 Area High Grade	809 Area	Remaining Reserves CAO	Remaining Reserves DCW&RWH
$450	$50	0.8 (80%)	$10.50	$2,543,640	$1,357,100	$6,409,400	$5,838,140
$450	$60	0.8 (80%)	$10.50	$1,979,640	$947,100	$4,949,400	$2,716,140
$450	$70	0.8 (80%)	$10.50	$1,415,640	$537,100	$3,489,400	($405,860)
$450	$50	0.8 (80%)	$11.50	$3,231,720	$1,316,100	$6,263,400	$5,525,940
$450	$60	0.8 (80%)	$11.50	$2,667,720	$906,100	$4,803,400	$2,403,940
$450	$70	0.8 (80%)	$11.50	$2,103,720	$496,100	$3,343,400	($718,060)
$450	$50	0.8 (80%)	$12.50	$3,175,320	$1,275,100	$6,117,400	$5,213,740
$450	$60	0.8 (80%)	$12.50	$2,611,320	$865,100	$4,657,400	$2,091,740
$450	$70	0.8 (80%)	$12.50	$2,047,320	$455,100	$3,197,400	($1,030,260)
$450	$50	0.85 (85%)	$10.50	$3,706,890	$1,596,950	$7,362,050	$7,383,530
$450	$60	0.85 (85%)	$10.50	$3,142,890	$1,186,950	$5,902,050	$4,261,530
$450	$70	0.85 (85%)	$10.50	$2,578,890	$776,950	$4,442,050	$1,139,530
$450	$50	0.85 (85%)	$11.50	$3,650,490	$1,555,950	$7,216,050	$7,071,330
$450	$60	0.85 (85%)	$11.50	$3,086,490	$1,145,950	$5,756,050	$3,949,330
$450	$70	0.85 (85%)	$11.50	$2,522,490	$735,950	$4,296,050	$827,330
$450	$50	0.85 (85%)	$12.50	$3,594,090	$1,514,950	$7,070,050	$6,759,130
$450	$60	0.85 (85%)	$12.50	$3,030,090	$1,104,950	$5,610,050	$3,637,130
$450	$70	0.85 (85%)	$12.50	$2,466,090	$694,950	$4,150,050	$515,130
$450	$50	0.9 (90%)	$10.50	$4,125,660	$1,836,800	$8,314,700	$8,928,920
$450	$60	0.9 (90%)	$10.50	$3,561,660	$1,426,800	$6,854,700	$5,806,920
$450	$70	0.9 (90%)	$10.50	$2,997,660	$1,016,800	$5,394,700	$2,684,920
$450	$50	0.9 (90%)	$11.50	$4,069,260	$1,795,800	$8,168,700	$8,616,720
$450	$60	0.9 (90%)	$11.50	$3,505,260	$1,385,800	$6,708,700	$5,494,720
$450	$70	0.9 (90%)	$11.50	$2,941,260	$975,800	$5,248,700	$2,372,720
$450	$50	0.9 (90%)	$12.50	$4,012,860	$1,754,800	$8,022,700	$8,304,520
$450	$60	0.9 (90%)	$12.50	$3,448,860	$1,344,800	$6,562,700	$5,182,520
$450	$70	0.9 (90%)	$12.50	$2,884,860	$934,800	$5,102,700	$2,060,520

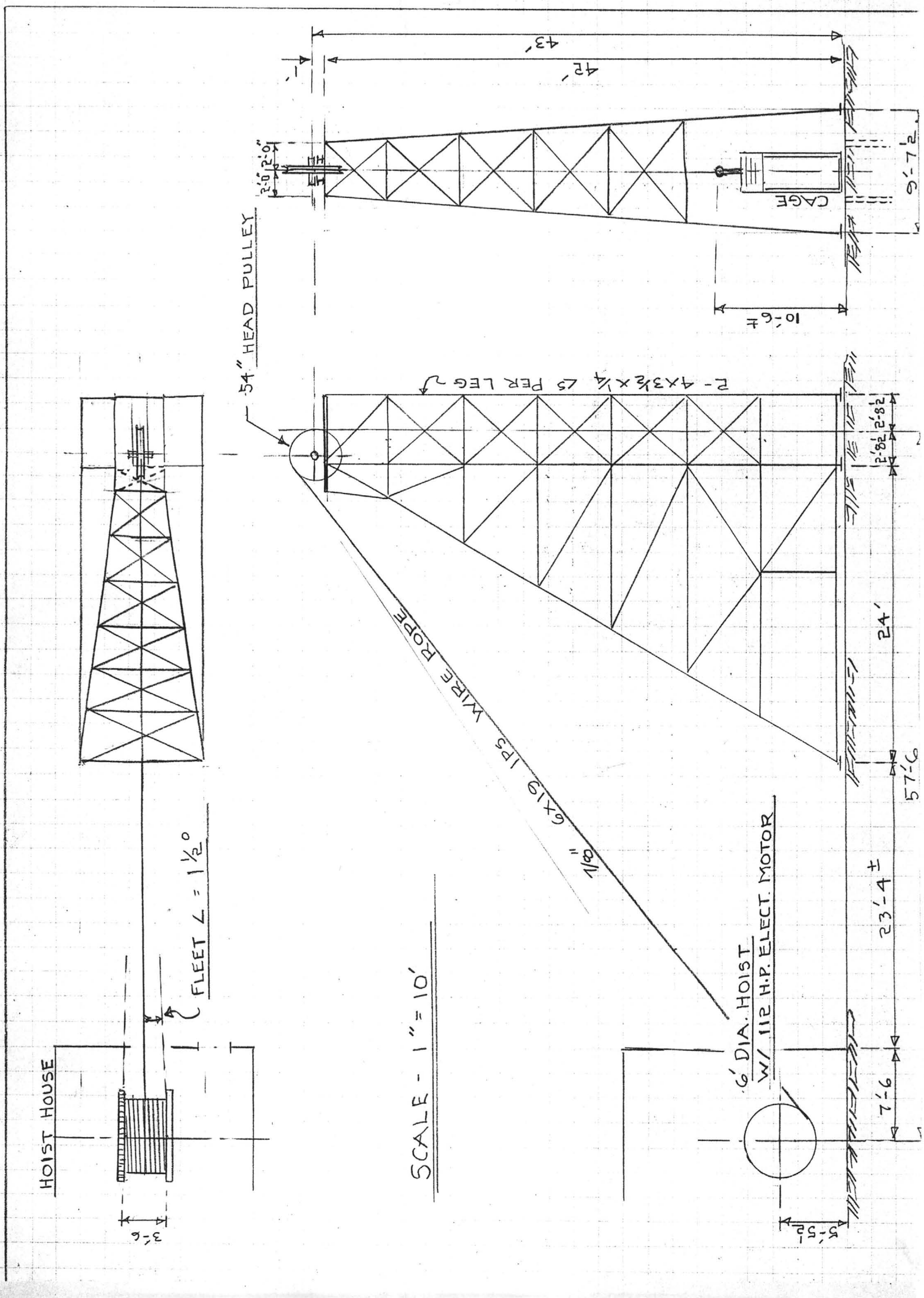
	Mining Cost per ton	Gold Recovery percent	Processing Cost per ton	Operating Profit			
				M-3 Area High Grade	809 Area	Remaining Reserves CAO	Remaining Reserves DCW&RWH
$500	$50	0.8 (80%)	$10.50	$3,288,120	$1,783,500	$8,103,000	$8,585,500
$500	$60	0.8 (80%)	$10.50	$2,724,120	$1,373,500	$6,643,000	$5,463,500
$500	$70	0.8 (80%)	$10.50	$2,160,120	$963,500	$5,183,000	$2,341,500
$500	$50	0.8 (80%)	$11.50	$3,976,200	$1,742,500	$7,957,000	$8,273,300
$500	$60	0.8 (80%)	$11.50	$3,412,200	$1,332,500	$6,497,000	$5,151,300
$500	$70	0.8 (80%)	$11.50	$2,848,200	$922,500	$5,037,000	$2,029,300
$500	$50	0.8 (80%)	$12.50	$3,919,800	$1,701,500	$7,811,000	$7,961,100
$500	$60	0.8 (80%)	$12.50	$3,355,800	$1,291,500	$6,351,000	$4,839,100
$500	$70	0.8 (80%)	$12.50	$2,791,800	$881,500	$4,891,000	$1,717,100
$500	$50	0.85 (85%)	$10.50	$4,497,900	$2,050,000	$9,161,500	$10,302,600
$500	$60	0.85 (85%)	$10.50	$3,933,900	$1,640,000	$7,701,500	$7,180,600
$500	$70	0.85 (85%)	$10.50	$3,369,900	$1,230,000	$6,241,500	$4,058,600
$500	$50	0.85 (85%)	$11.50	$4,441,500	$2,009,000	$9,015,500	$9,990,400
$500	$60	0.85 (85%)	$11.50	$3,877,500	$1,599,000	$7,555,500	$6,868,400
$500	$70	0.85 (85%)	$11.50	$3,313,500	$1,189,000	$6,095,500	$3,746,400
$500	$50	0.85 (85%)	$12.50	$4,385,100	$1,968,000	$8,869,500	$9,678,200
$500	$60	0.85 (85%)	$12.50	$3,821,100	$1,558,000	$7,409,500	$6,556,200
$500	$70	0.85 (85%)	$12.50	$3,257,100	$1,148,000	$5,949,500	$3,434,200
$500	$50	0.9 (90%)	$10.50	$4,963,200	$2,316,500	$10,220,000	$12,019,700
$500	$60	0.9 (90%)	$10.50	$4,399,200	$1,906,500	$8,760,000	$8,897,700
$500	$70	0.9 (90%)	$10.50	$3,835,200	$1,496,500	$7,300,000	$5,775,700
$500	$50	0.9 (90%)	$11.50	$4,906,800	$2,275,500	$10,074,000	$11,707,500
$500	$60	0.9 (90%)	$11.50	$4,342,800	$1,865,500	$8,614,000	$8,585,500
$500	$70	0.9 (90%)	$11.50	$3,778,800	$1,455,500	$7,154,000	$5,463,500
$500	$50	0.9 (90%)	$12.50	$4,850,400	$2,234,500	$9,928,000	$11,395,300
$500	$60	0.9 (90%)	$12.50	$4,286,400	$1,824,500	$8,468,000	$8,273,300
$500	$70	0.9 (90%)	$12.50	$3,722,400	$1,414,500	$7,008,000	$5,151,300

Gold Price per ounce	Mining Cost per ton	Gold Recovery percent	Processing Cost per ton	Operating Profit: M-3 Area High Grade	809 Area	Remaining Reserves CAO	Remaining Reserves DCW&RWH
$400	$50	0.8 (80%)	$10.50	($3,412,200)	$930,700	$4,715,800	$3,090,780
$400	$60	0.8 (80%)	$10.50	($3,976,200)	$520,700	$3,255,800	($31,220)
$400	$70	0.8 (80%)	$10.50	($4,540,200)	$110,700	$1,795,800	($3,153,220)
$400	$50	0.8 (80%)	$11.50	$2,487,240	$889,700	$4,569,800	$2,778,580
$400	$60	0.8 (80%)	$11.50	$1,923,240	$479,700	$3,109,800	($343,420)
$400	$70	0.8 (80%)	$11.50	$1,359,240	$69,700	$1,649,800	($3,465,420)
$400	$50	0.8 (80%)	$12.50	$2,430,840	$848,700	$4,423,800	$2,466,380
$400	$60	0.8 (80%)	$12.50	$1,866,840	$438,700	$2,963,800	($655,620)
$400	$70	0.8 (80%)	$12.50	$1,302,840	$28,700	$1,503,800	($3,777,620)
$400	$50	0.85 (85%)	$10.50	$2,915,880	$1,143,900	$5,562,600	$4,464,460
$400	$60	0.85 (85%)	$10.50	$2,351,880	$733,900	$4,102,600	$1,342,460
$400	$70	0.85 (85%)	$10.50	$1,787,880	$323,900	$2,642,600	($1,779,540)
$400	$50	0.85 (85%)	$11.50	$2,859,480	$1,102,900	$5,416,600	$4,152,260
$400	$60	0.85 (85%)	$11.50	$2,295,480	$692,900	$3,956,600	$1,030,260
$400	$70	0.85 (85%)	$11.50	$1,731,480	$282,900	$2,496,600	($2,091,740)
$400	$50	0.85 (85%)	$12.50	$2,803,080	$1,061,900	$5,270,600	$3,840,060
$400	$60	0.85 (85%)	$12.50	$2,239,080	$651,900	$3,810,600	$718,060
$400	$70	0.85 (85%)	$12.50	$1,675,080	$241,900	$2,350,600	($2,403,940)
$400	$50	0.9 (90%)	$10.50	$3,288,120	$1,357,100	$6,409,400	$5,838,140
$400	$60	0.9 (90%)	$10.50	$2,724,120	$947,100	$4,949,400	$2,716,140
$400	$70	0.9 (90%)	$10.50	$2,160,120	$537,100	$3,489,400	($405,860)
$400	$50	0.9 (90%)	$11.50	$3,231,720	$1,316,100	$6,263,400	$5,525,940
$400	$60	0.9 (90%)	$11.50	$2,667,720	$906,100	$4,803,400	$2,403,940
$400	$70	0.9 (90%)	$11.50	$2,103,720	$496,100	$3,343,400	($718,060)
$400	$50	0.9 (90%)	$12.50	$3,175,320	$1,275,100	$6,117,400	$5,213,740
$400	$60	0.9 (90%)	$12.50	$2,611,320	$865,100	$4,657,400	$2,091,740
$400	$70	0.9 (90%)	$12.50	$2,047,320	$455,100	$3,197,400	($1,030,260)

UVX PROJECT ~ JEROME, AZ.

EDITH HEADFRAME & HOIST

MAY 198



Rev. - Mar. 87
~~May 31, 8~~

DMEA ~ GEN'L NOTES

A) Skip Calculations

1) Skip ore capacity = 2 Tons
2) Production capacity
   a) Hoisting hours per 8 hr. shift = 6.0 hrs
   b) Trips = 100 Ton/shift ÷ 6.0 hrs = 16 2/3 tons/hr.
      = 8 2/3 cycles per hour @ 2.0 tons/cycle
      = 7.2 minutes/cycle
3) Cycle Time
   a) Skip pocket to Dump Position = 800 + 50 + 30 = 880 ft
   b) Creep time out of load pocket = 4 sec
   c) " " into dump pocket = 8 sec
   d) Load time = 15 sec
   e) Dump time = 15 sec
   f) Hoisting time

$$t_2 = \frac{h - 2s}{v_1}$$

$$v = \frac{a h v_1}{v_1^2 + a h}$$

t_2 = time to max speed (sec)
h = hoisting distance (ft)
s = dist passed over during accel. (ft)
v_1 = Max Velocity (ft/sec)
a = accel. (ft/sec²)
t = hoisting time (sec)
t_1 = time to accelerate (sec.)
v = Mean Velocity (ft/sec)

   g) Hoisting speed = 290 fpm = 4.8 ft/sec
   h. Time to accelerate = 4 sec.
   i) " " decelerate = 4 sec.
   j) $s = \frac{1}{2} a t^2 = \frac{1}{2} \times 5\ \text{ft/sec}^2 (4\ \text{sec})^2 = \frac{1}{2} \times 5 \times 16 = 40\ \text{ft}$
   k) hoisting height @ 290 ft/min = 880 ft − 80' = 800 ft
   l) ∴ hoisting time @ 290 ft/min = 800 ft ÷ 4.8 ft/sec = 166 sec
      = 2 min : 46 sec.
   m) Hoisting cycle

(could be lowered @ faster rate)

   = Hoisting time plus lowering time
   = (15 + 4 + 166 + 8 + 15) sec + (14 + 166 + 4)
   (Load, (a), (hoist), slow, dump)
   = 208 sec + 184 sec = 392 sec
   = 6.5 min = 6 1/2 min.

4) Hoisting load
   a) Man cage = 1,500
   b) Skip 3'W x 5' Long x 5' high = 3,500
   c) 7/8" ϕ rope @ 850 ft @ 1.23 #/ft = 1,100
   d) Ore load @ 2 Ton = 4,000 #

Dead wt. = 10,100 #

= 5.1 Ton

UVX ~ HOIST MOTOR CALCULATIONS

3/87

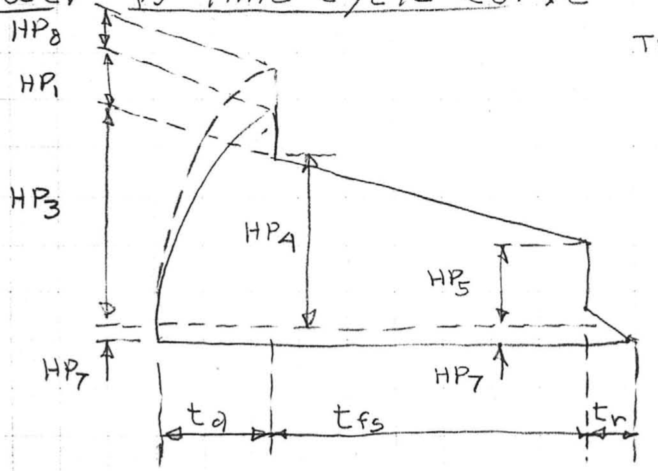
I DATA:

1. Static Load 7/8 ϕ Rope @ 880ft @ 1.23 #/ft = 1,100.#
2. Cage Wt. (Est.) = 1,500#
3. Skip Wt. (Est) = 3,500#
4. Pay Load = 4,000#

Tot. Est. - 10,100#
= 5.05 Ton

5. Hoist Speed = 300 ft/min = 5.0 fps

II Power Vs Time Cycle Curve



TSL = Total Load and mass hoist rotating Par

V = Rope Speed (fps) = 5.0 fps

Ta = Tot. Acceleration time (Creep plus creep to full speed) = 2sec + 4sec = 6s

R = Rope Wt. = 1,100#

SL = Skip Load = 4,000#

SW = Dead Wt. of skip & Cage = 5,000#

L_M = Max. Static Load = 9,600# = 5.01 To

III Power Calculations

1. $H.P._1 = \dfrac{TSL \times V^2}{32.2 \times T_a \times 550} = \dfrac{38{,}300 \times 5.0^2}{32.2 \times 6 \times 550} = \underline{9.01\ HP}$. (H.P. Req'd to accelerate)

   [TSL = EEW (Page 15-17/SME Hdbk) + SL + 2SW + 2R = 22,000 + 4,000 + 2 (4,500#) + 2 (1,100) = 22,000 + 4,000 + 10,100 + 2,200 = $\underline{38{,}300}$]

2. $H.P._3$ = Running Horse Power at Bottom $= \dfrac{L_M \times V}{550} = \dfrac{10{,}100\# \times 5\ ft/sec}{550} = \underline{91.82\ HP}$

3. HP_7 = Correction for gear and motor efficiency being less than 100% $= \dfrac{SL \times V}{550} \times 0.17$
   $= \dfrac{4{,}000 \times 5.0}{550} \times 0.176 = \underline{6.40\ H.P.}$

4. Peak HP = $HP_1 + HP_3 + HP_7$ = 9.01 + 91.82 + 6.40 = $\underline{107.2\ H.P.}$

5. HP of Existing Hoist = $\underline{112\ H.P.}$

WIRE ROPE CALCULATIONS

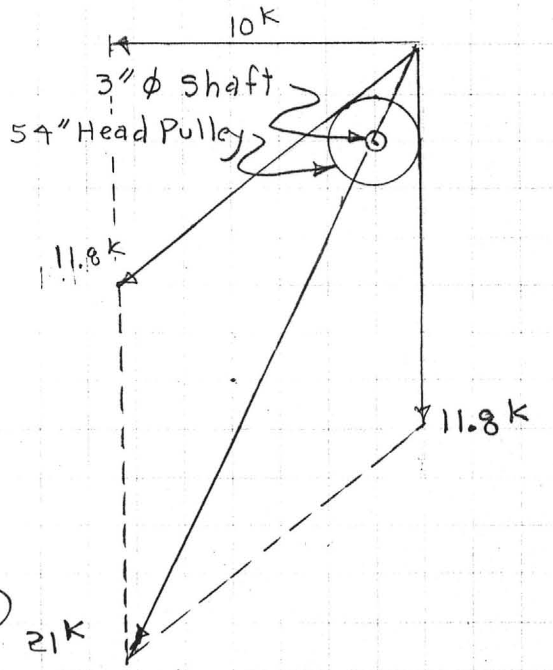
1) Wire Rope $7/8\ \phi$, W/t = 1.23 #/ft, Hoist length - 880'
2) Safety Factor = 7.0 U.S. & 7.7 Canadian
   (P. 15-5 SME Hdbk)
3) Total Static Load on Rope (Max.) = 10,100#
4) Rope Breaking Strengths
   a) From Martensen Enterprises (Roger Voorhees) TUC. 884-5846
      - Ex. Improved Plow Steel
        1" ϕ B.S. = 51.7 tons Cost Delivered = 1.81\$/ft
        $7/8\ \phi$ B.S. = 39.8 Tons Cost Delivered = 1.49\$/ft
   b) Wire Rope Hdbk. (Published 1946)
      Breaking Strength in tons, 6X19 Hoisting Rope

Size	Monitor Steel	Plow Steel	Mild Plow Steel	Wt./Ft
$7/8\ \phi$	32.2	28.0	24.3	1.23
1 ϕ	41.8	36.4	31.6	1.60

5) Req'd Min. Breaking Strength for Load of 10,100#
   = 5.05 tons × 7.0 s.f. = 35.35 Tons.

6) Existing Edith Shaft 7/8 hoist rope was tested to breaking strength by Martensen Inc. of 9/23/86 and broke @ 75,600# = 37.8 Tons.
   Was Clasified as 6X19 FC RLL (Fiber Core, Reg. Lang Lay?)

HEAD PULLEY FORCES



1. Static Load = 10.1 K
2) Acc'l Load = 107.2 HP =

$$F = \frac{33{,}000\ \text{ft-\#/Min/HP} \times 107.2\ \text{HP}}{300\ \text{ft/min}} = 11.8^{K}$$

3) Shaft Shear Strength.

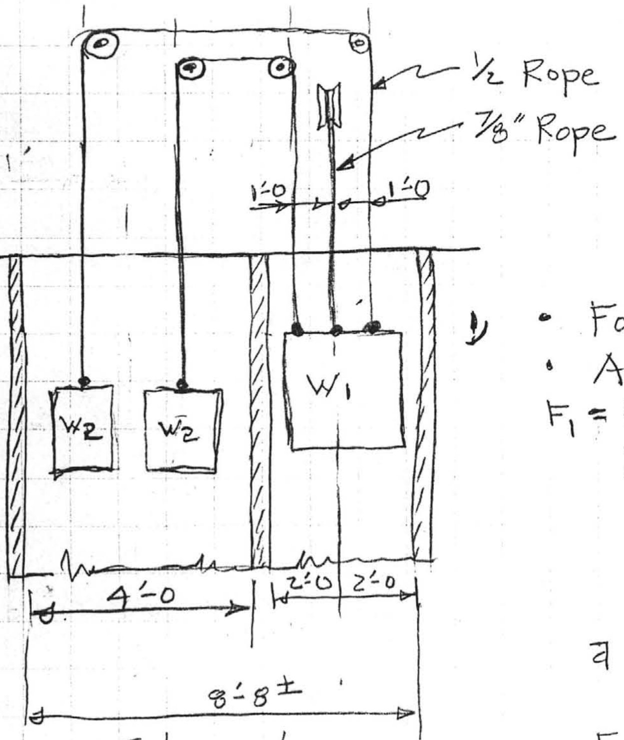
$$\frac{\pi d^2}{4} = \frac{\pi \times 3^2}{4} = 7.07\ \square''$$

$$S_s = \frac{21{,}000^{\#}}{7.07\ \square''} \times 50\% = 1{,}485\ \#/\square''$$

$$S_s\ (\text{allowable}) = 0.4\,F_y = 0.4\,(30{,}000) = 12{,}000\ \text{psi}$$

I Hoisting Rope Considerations.

A) Scheme #1



Schematic

W_1 = Skip, rope & cage load in kips
W_2 = Counteweight load in kips
s = Distance in feet
V = Rope velocity = 300 fpm = 5 fps
a = Linear acceleration in fps
g = acceleration (gravity) = 32.2 fpse
t = Time of acceleration in seconds
F = Force in kips

1) • Force required to accelerate a load W
• Accelerate load to 5 fps in 4 seconds

F_1 = Force to accelerate load W_1 of 12^k and counterweight loads W_2 of 3^k eac.

∴ Effective hoist rope load max = 6^k

∴ $W_{HR} = 12^k - 6^k = 6^k$

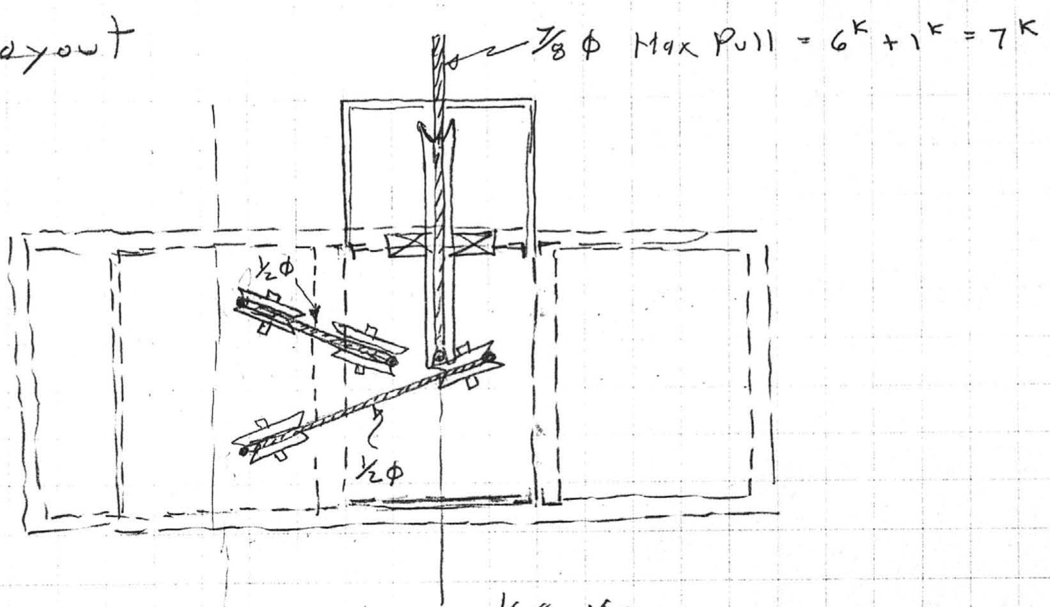
$$a = \frac{V}{t} = \frac{5\,ft/sec}{5\,sec} = 5\,ft/sec^2$$

$$F_1 = \frac{W_{HR}}{g} \times \frac{V}{t} = \frac{6^k}{32.2\,ft/sec^2} \times 5\,ft/sec^2 = \frac{30}{32.2} = 1.0^k$$

2) Running H.P. $= \frac{Lm \times V}{550} = \frac{6{,}000\# \times 5\,ft/sec}{550} = 55$ H.P.

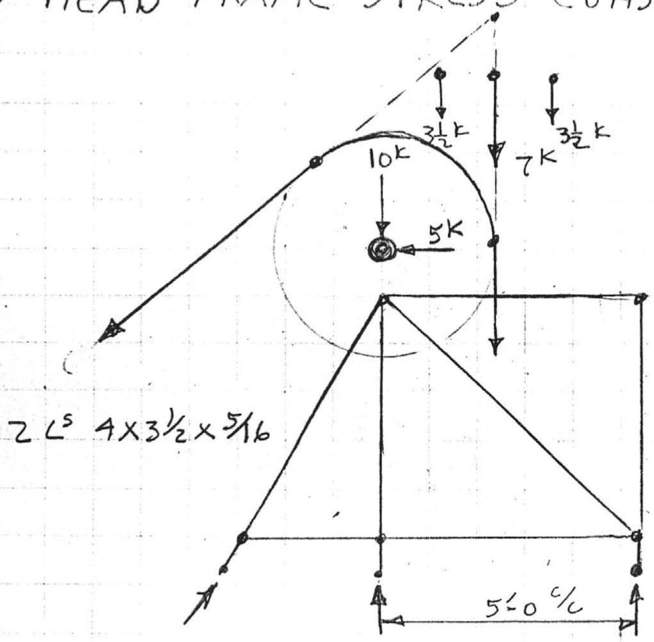
3) Accelerating HP $= \frac{1{,}000\# \times 5\,ft/sec}{550} = 9$ H.P.

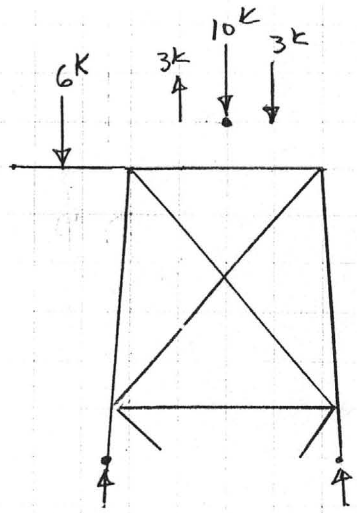
B) Top Layout



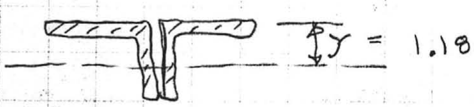
1/4" = 1'-0

C/ Head Frame Stress Considerations

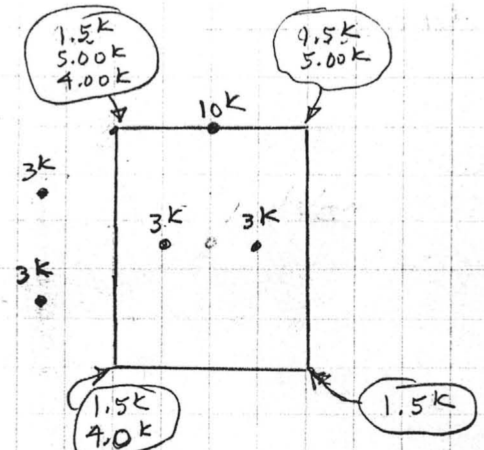




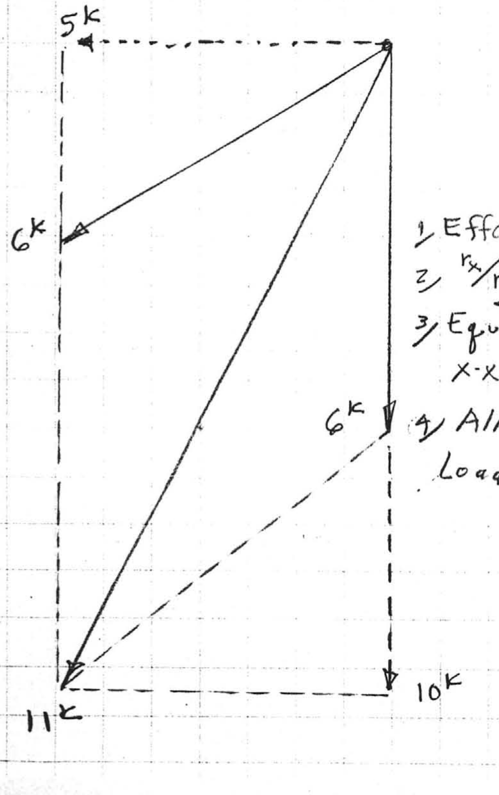
- Combined Vert Load = 24k
- Area 2 ∠s 4×3½×5/16 = 4.49 □"



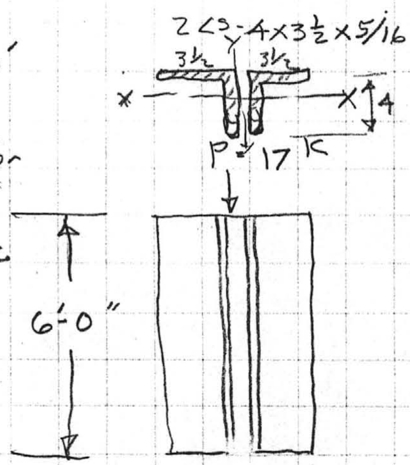
I = 7.12 in⁴ r = 1.26"
Wt = 15.4 #/ft



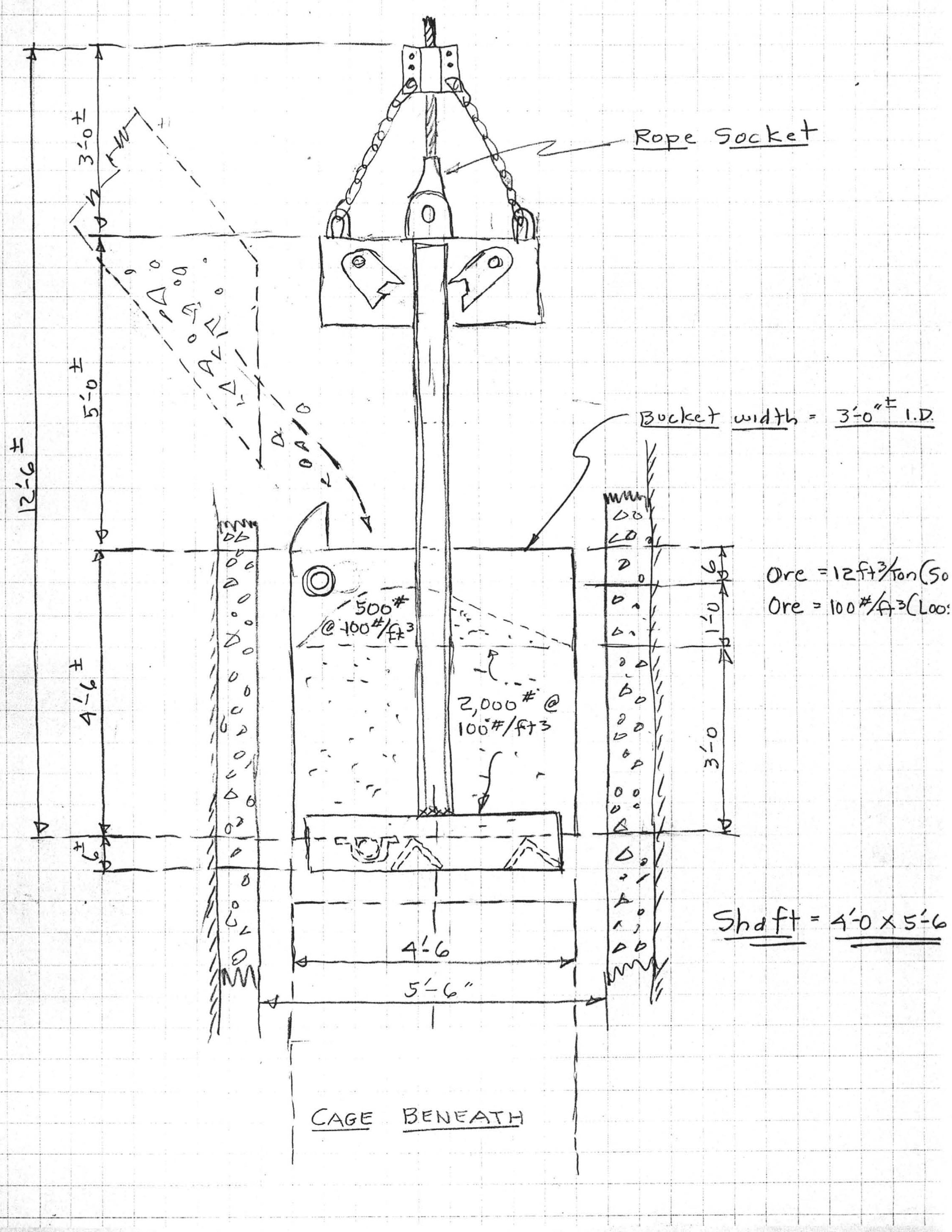
Max. Col. Load = 10.5k + 1.5k = 12k static
Static = 12k + Dynamic = 5k = 17k



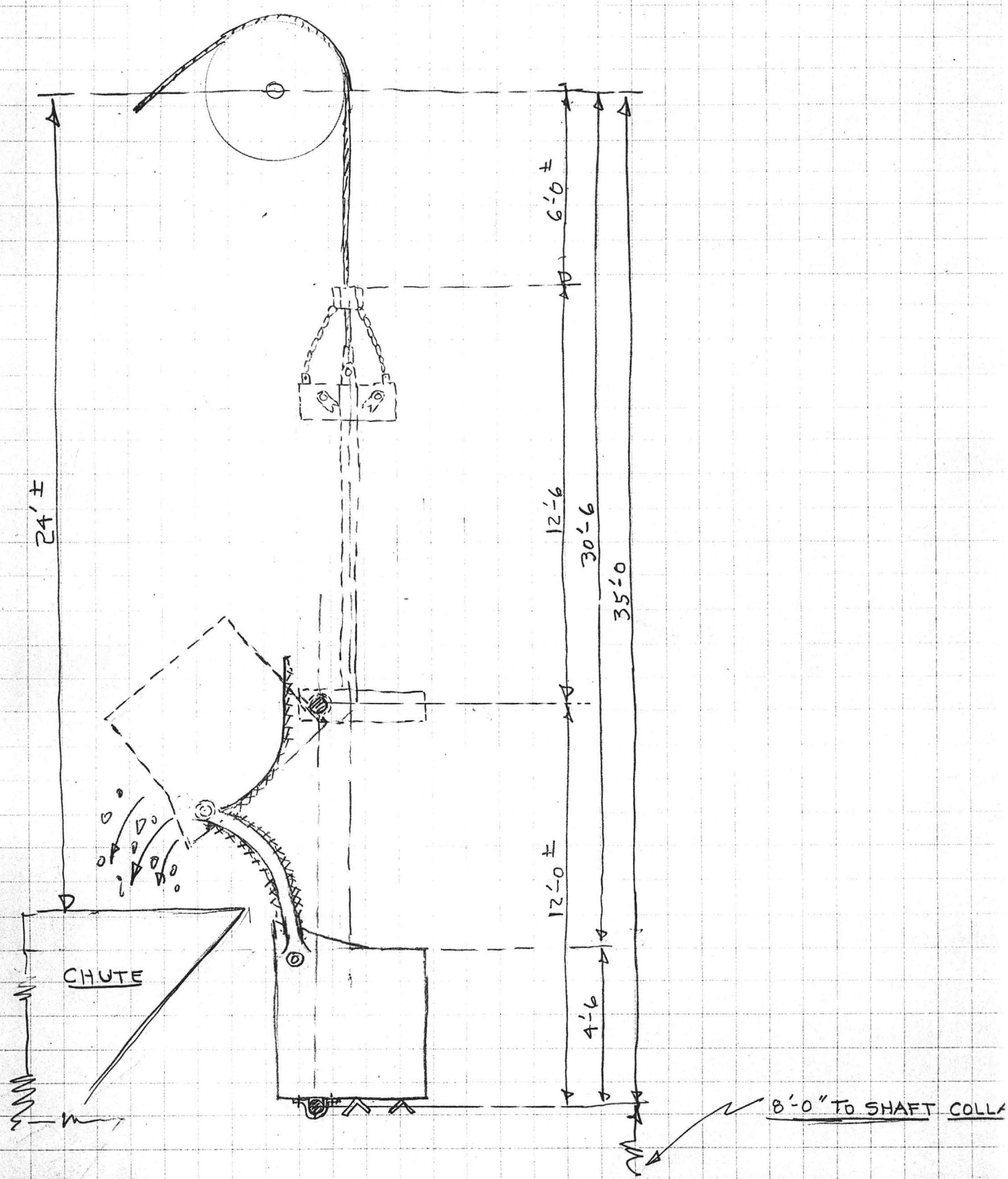
1. Effective length = 6'
2. $r_x / r_y = \frac{1.26}{1.55} = 0.81$
3. Equivalent Length for X-X axis = $\frac{6.0'}{0.81} = 7.4'$
4. Allowable Concentric Load = 79k



SKIP CONSIDERATIONS

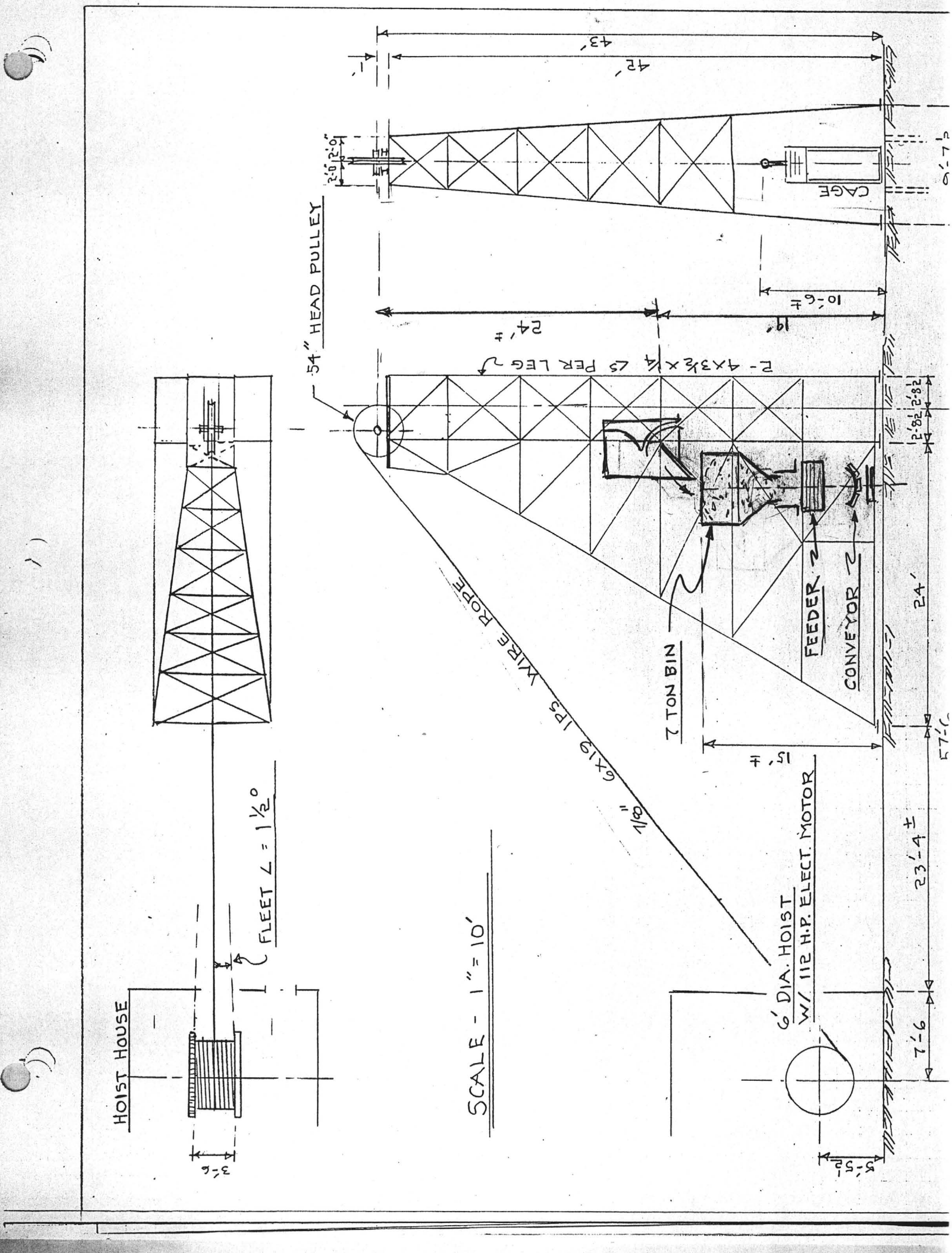


SKIP



EDITH HEADFRAME & HOIST

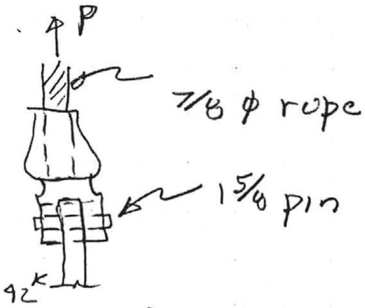
MAY 19



UVX 5/24/87

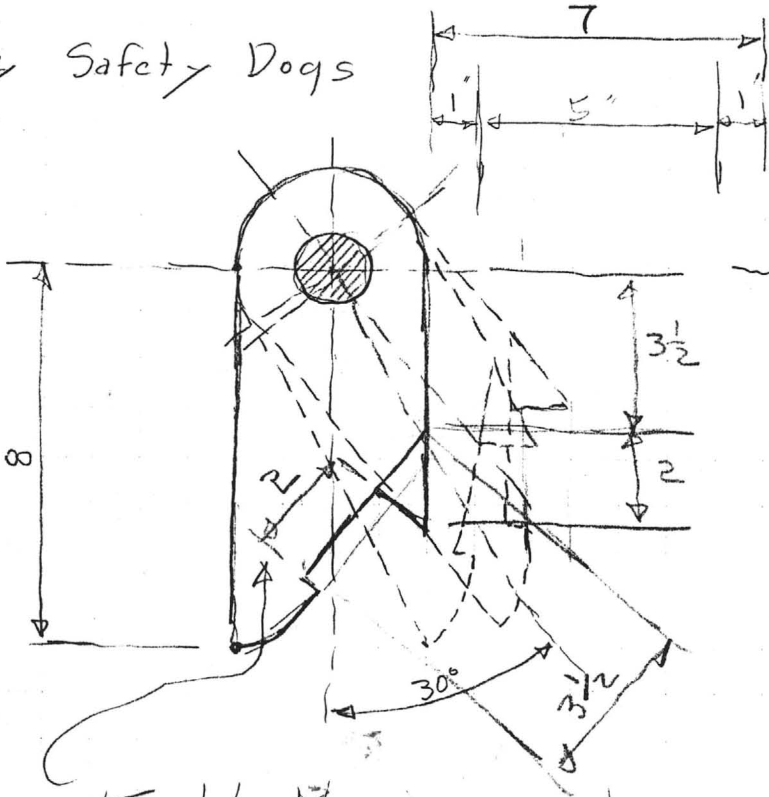
Misc. Calculations

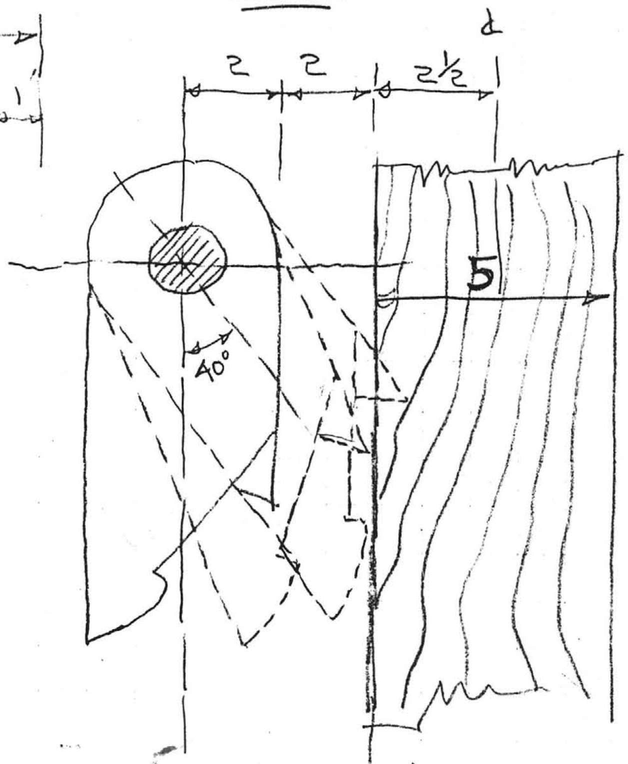
1) Skip support cable pin



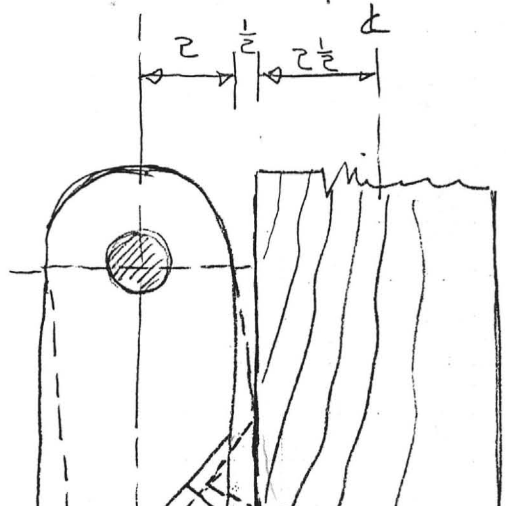
- Area $1\frac{5}{8}\,\phi$ pin = 2.07 □"
- Allowable shear stress = 6,000#
- Allowable "P" single shear = $6^k \times 2.07 = 12.42^k$
- Doube shear = $2 \times 12.4^k = 24.8^k$ (O.K.)

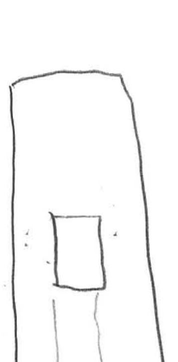
2) Safety Dogs



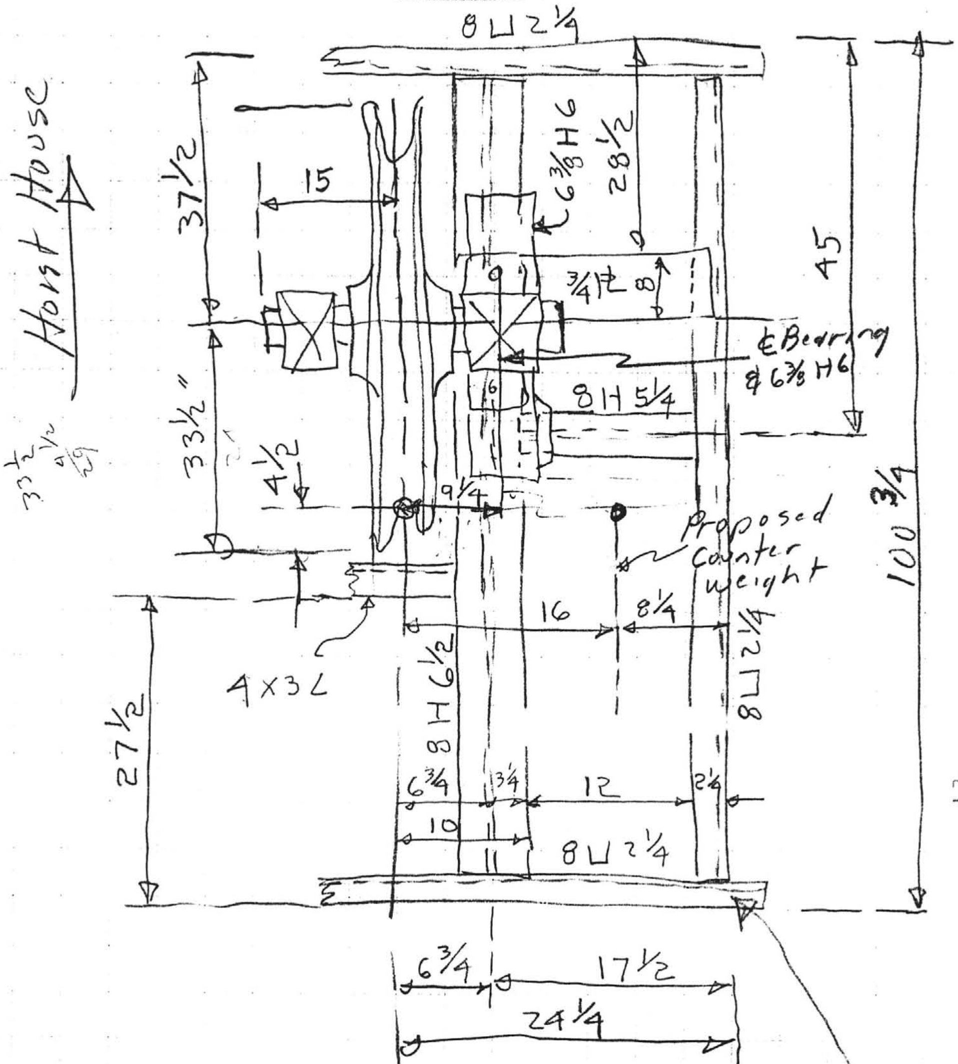


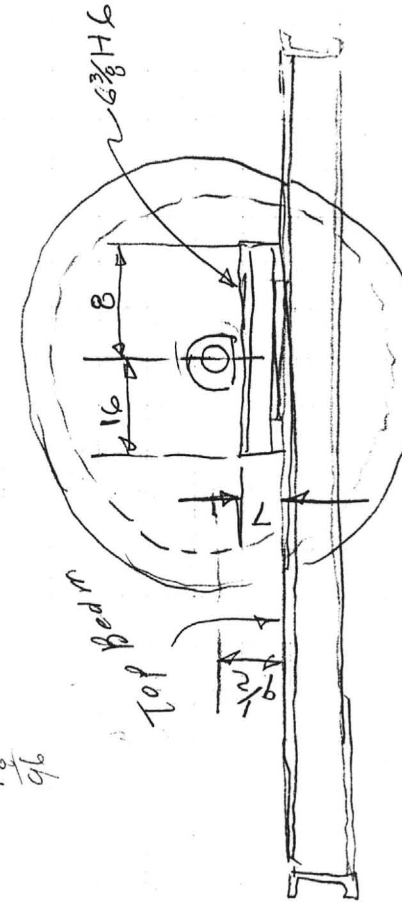
Field Measurements





UVX



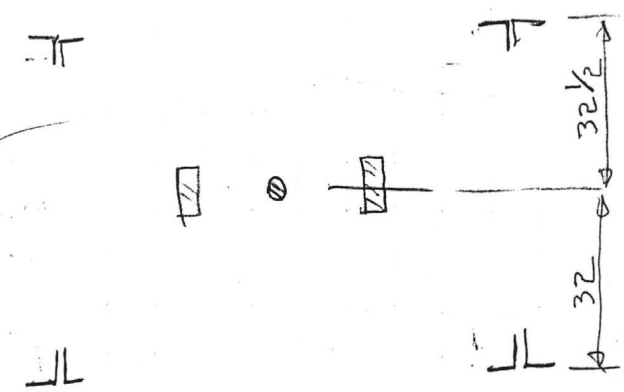


23 5/4
17 2/4
6 3/4

Note: 42'-2"
From top to
Ground (Bot of
base plates)

0 - 9
16 - 0
16 - 0
9'-5
42-2"

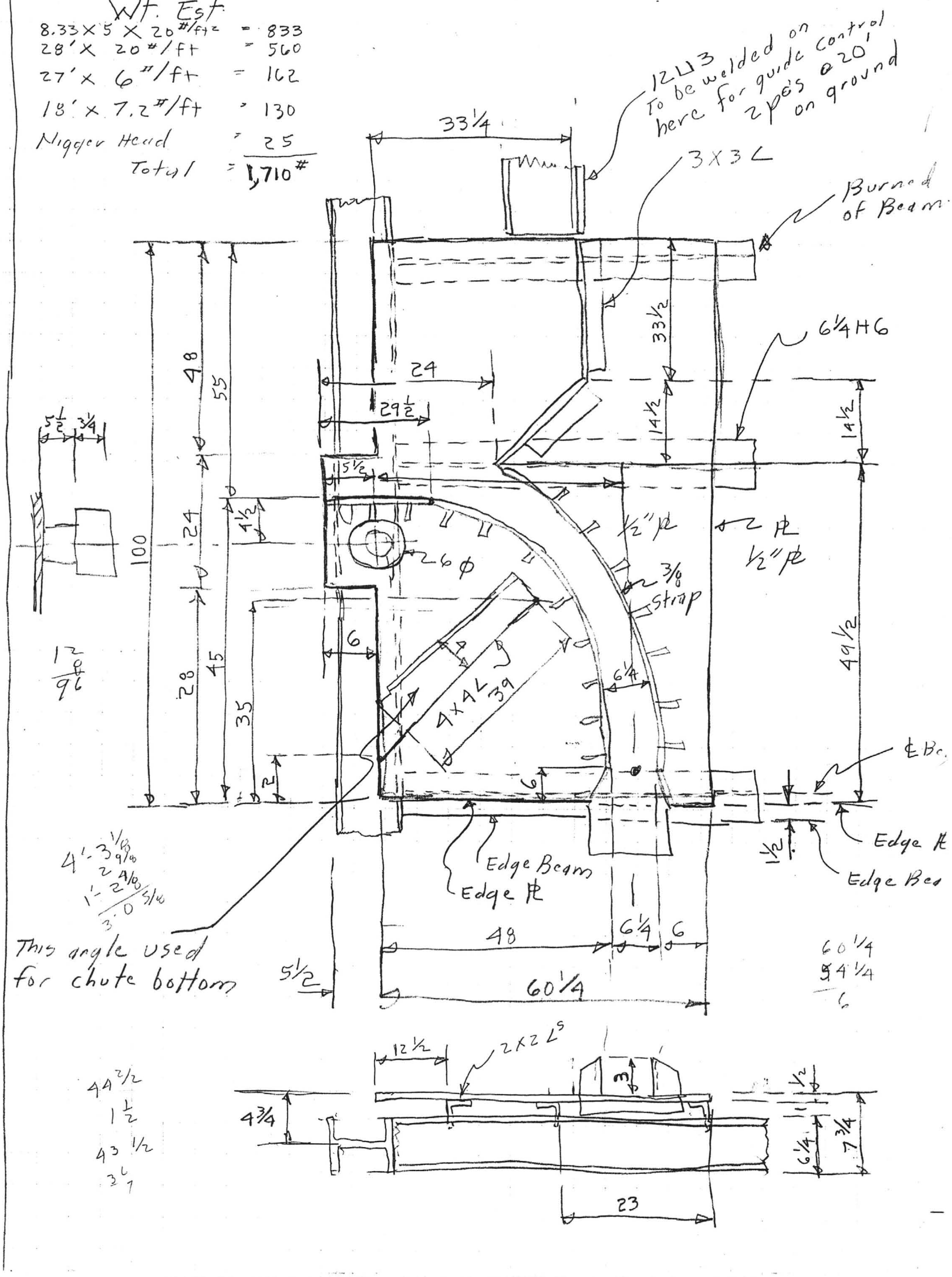
24¼
12¼
12



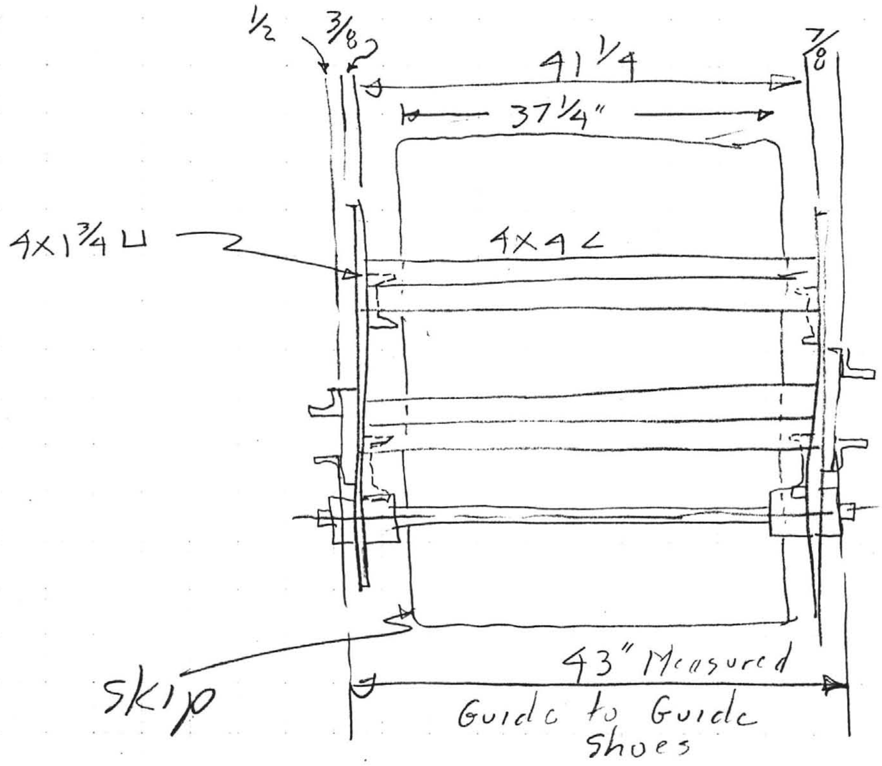
UVX

Wt. Est.

8.33 × 5 × 20#/ft²	=	833
28′ × 20#/ft	=	560
27′ × 6#/ft	=	162
18′ × 7.2#/ft	=	130
Nigger Head	=	25
Total	=	1,710#

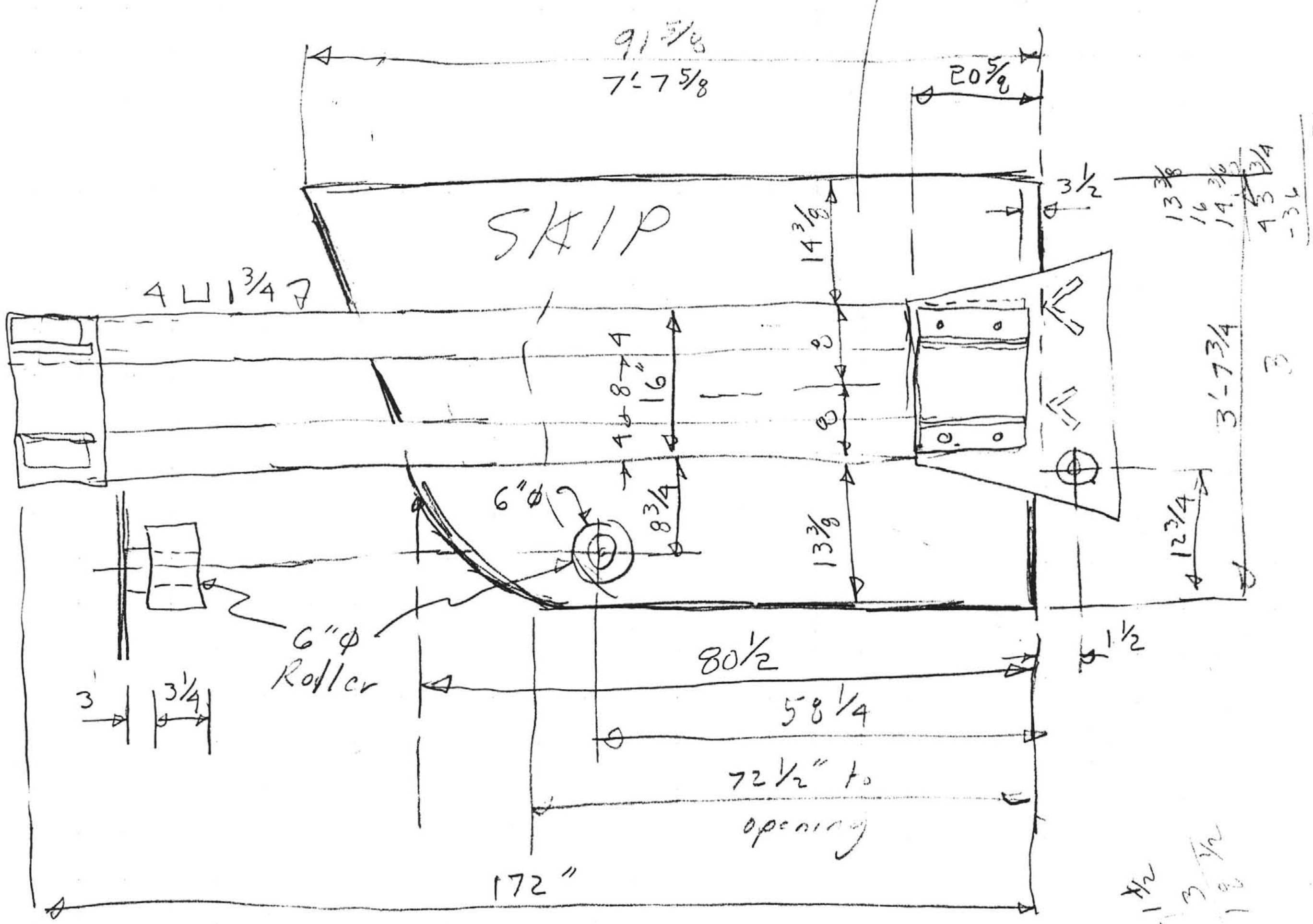


UYX

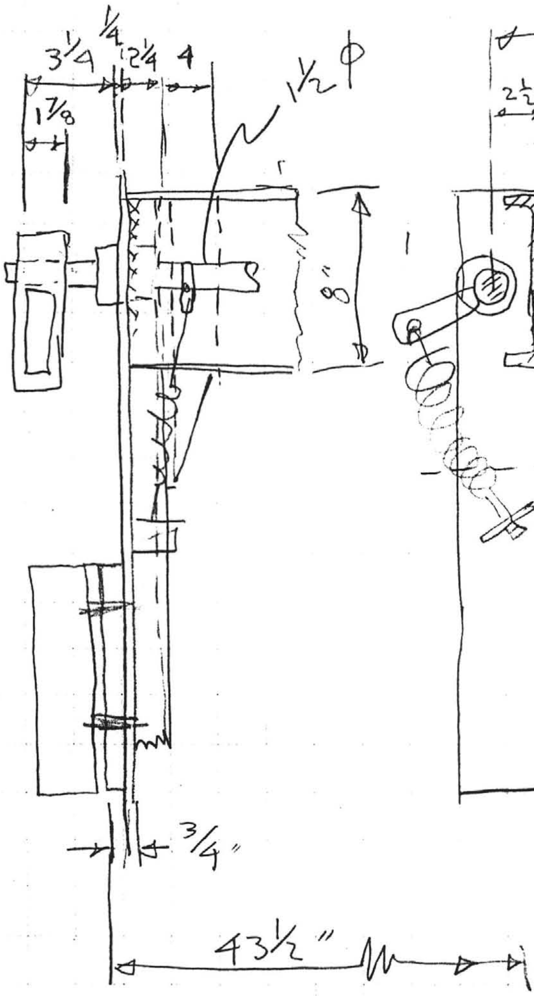


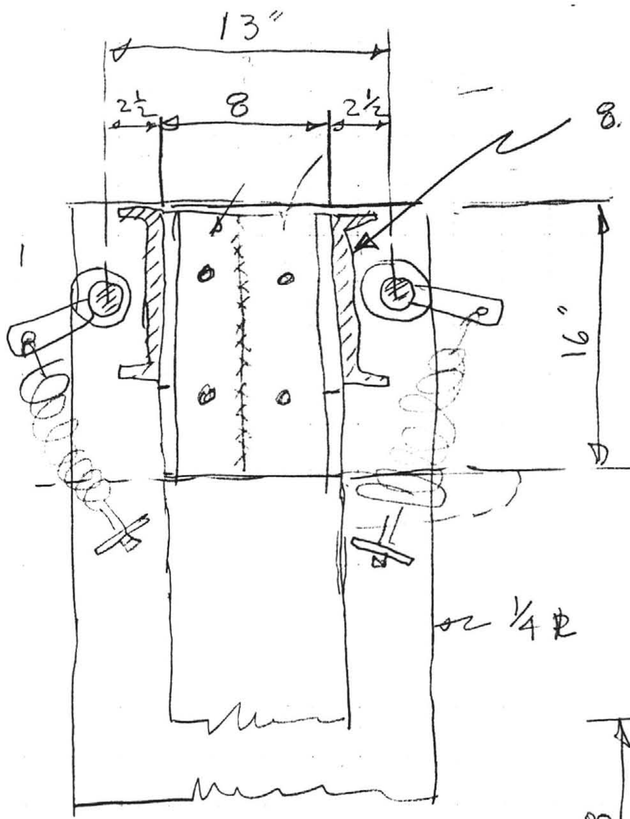
41 1/4
37 1/4
4"

vol = 3.6' x 2.83 x 5'
= 51 ft³
@ 105#/ft³ = 2.7 To.



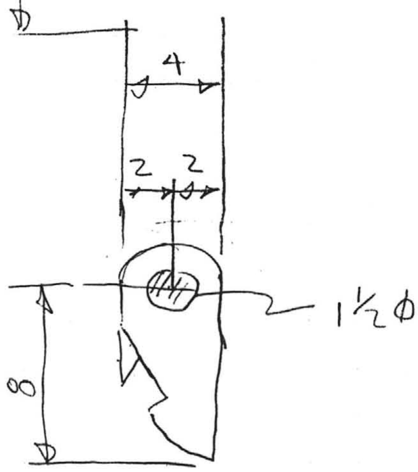
UVX



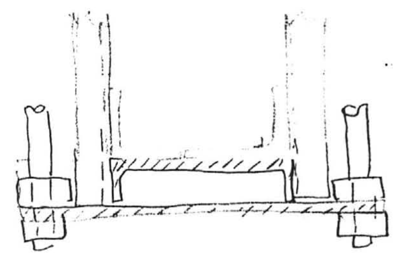


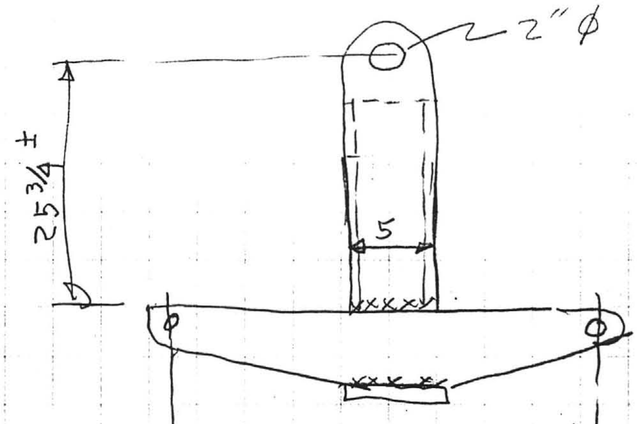
13
5

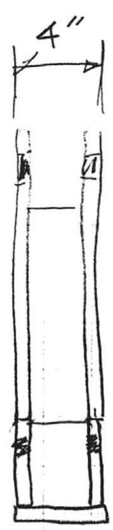
10½
5
5½



Safety Dog

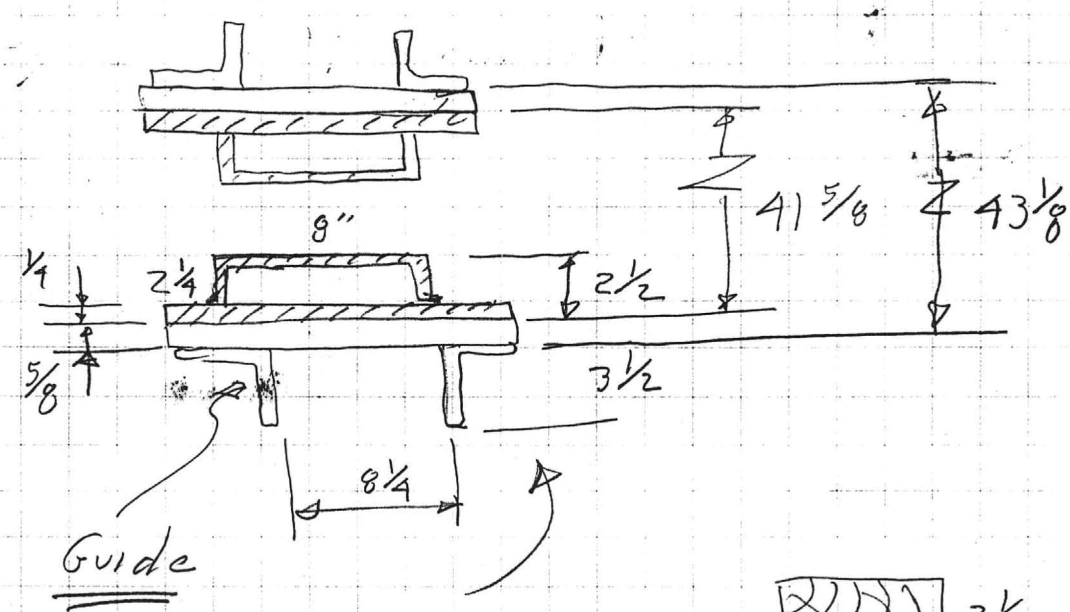




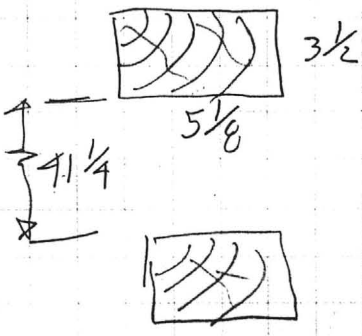


UVX

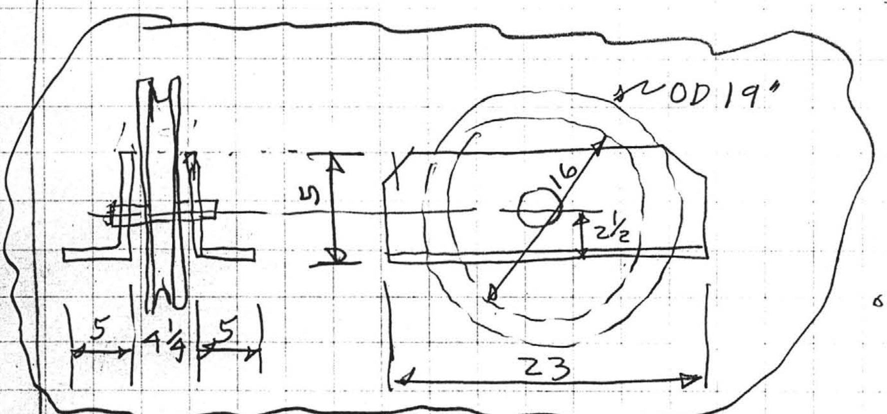
41 5/8
1 2/8
42 7/8



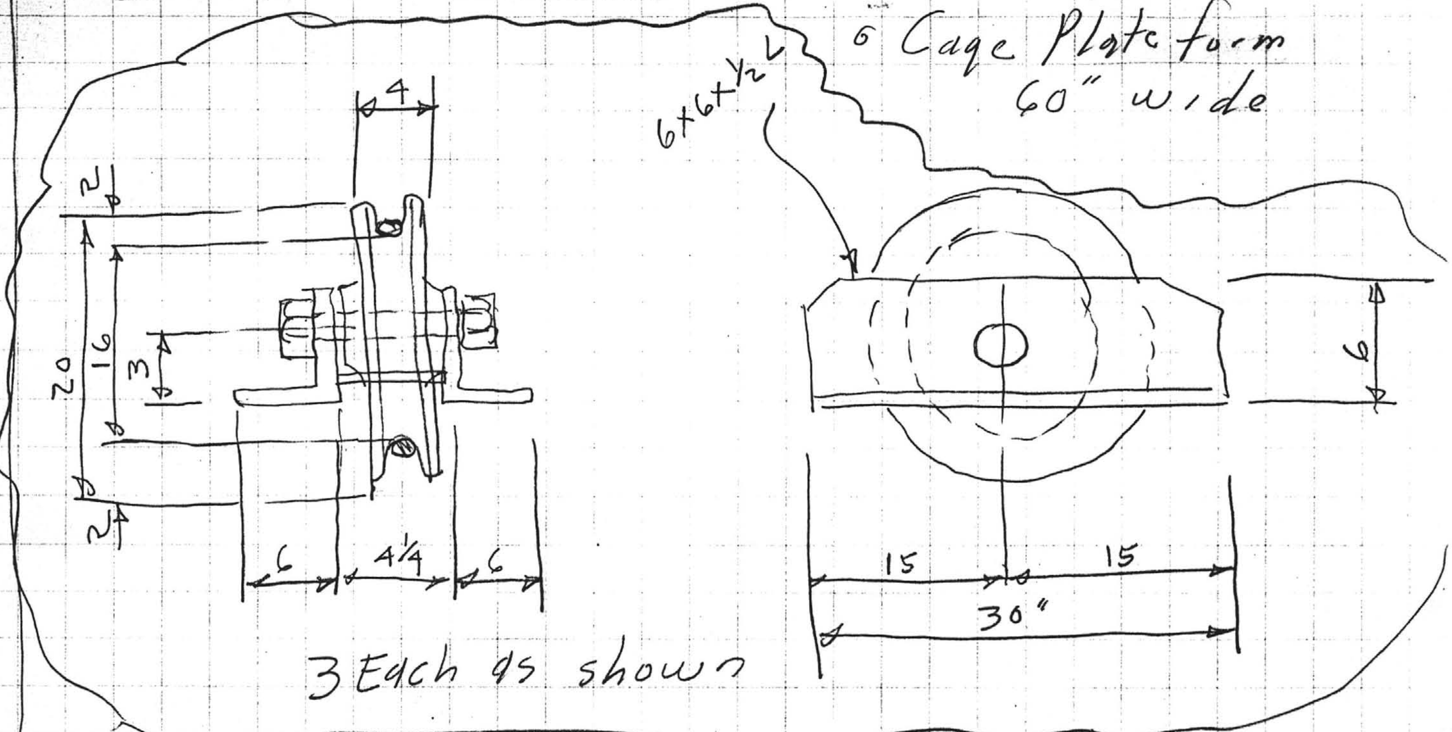
To Safety Latches



Existing Shaft



- Make new skip 40" wide to guide wear plates
- Cage Plate form 60" wide



3 Each as shown

/20/87

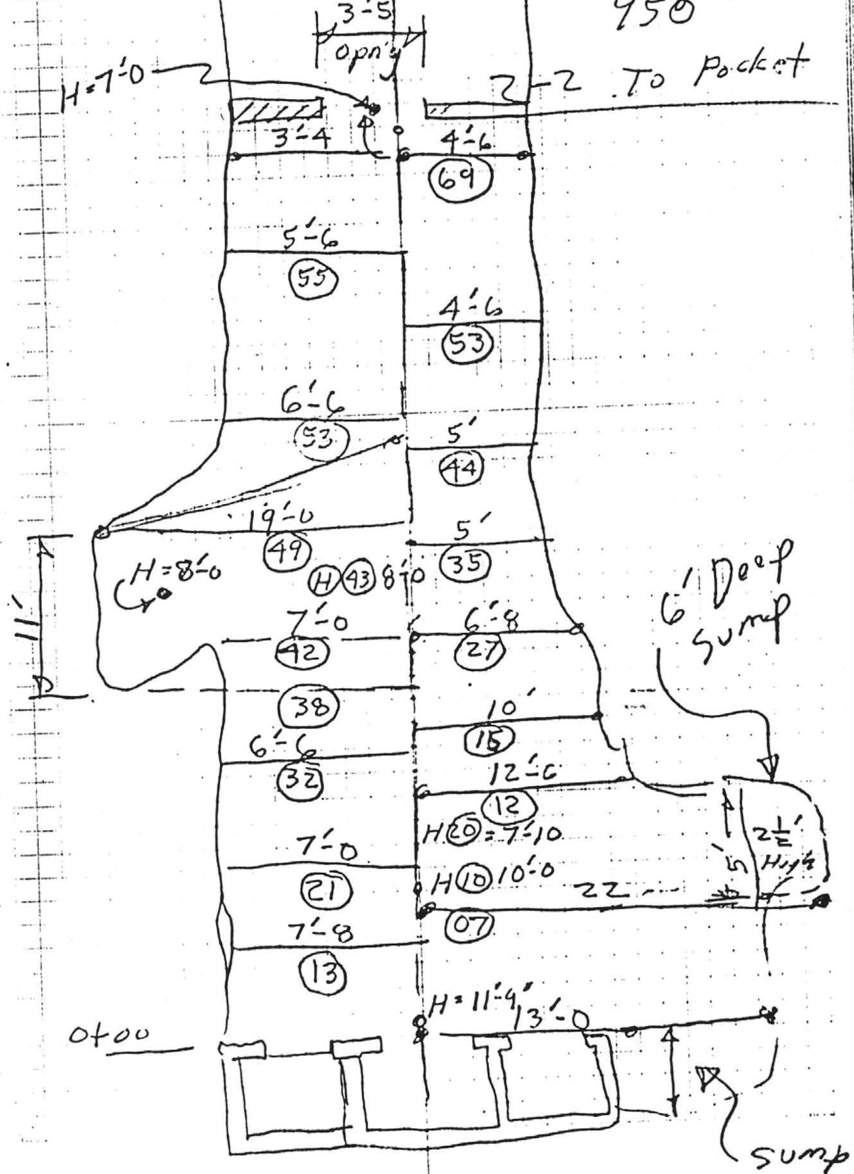
UVX - 5/20/87

H.K., K.K., Joe Fernandez

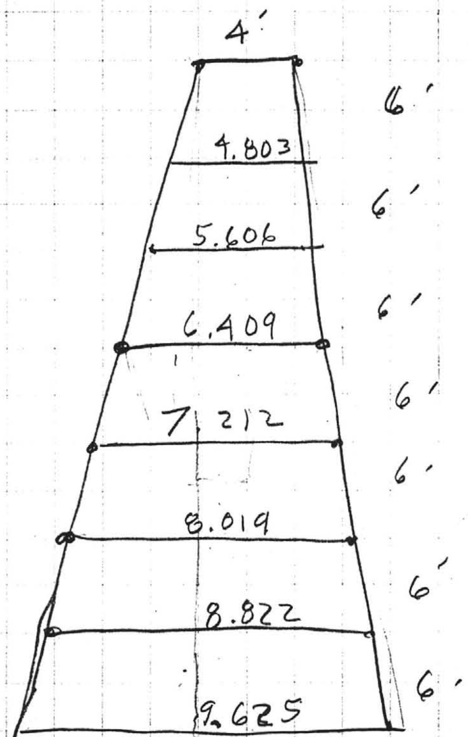
1-2-3-1

1-5-3-2

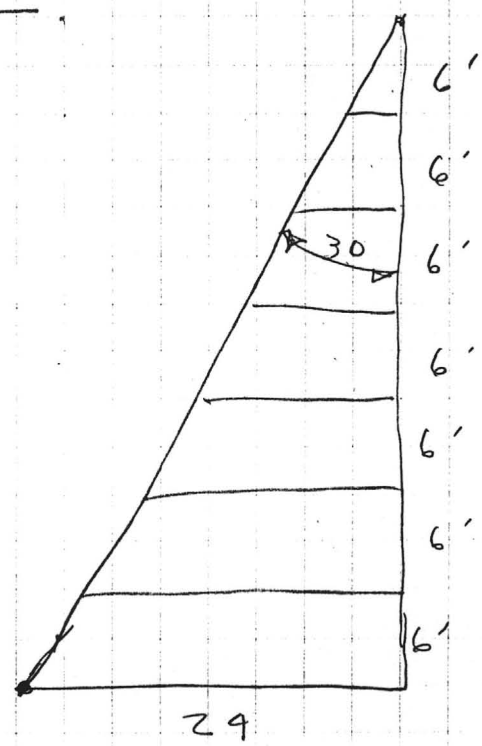
950



UVX ~ Jerome 6/1/87



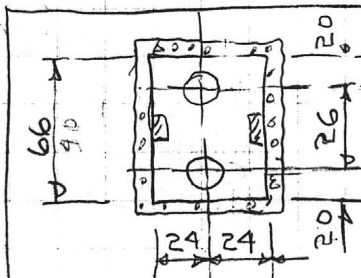
$$\begin{array}{r} 9.625 \\ 4.000 \\ \hline 5.625 \end{array}$$



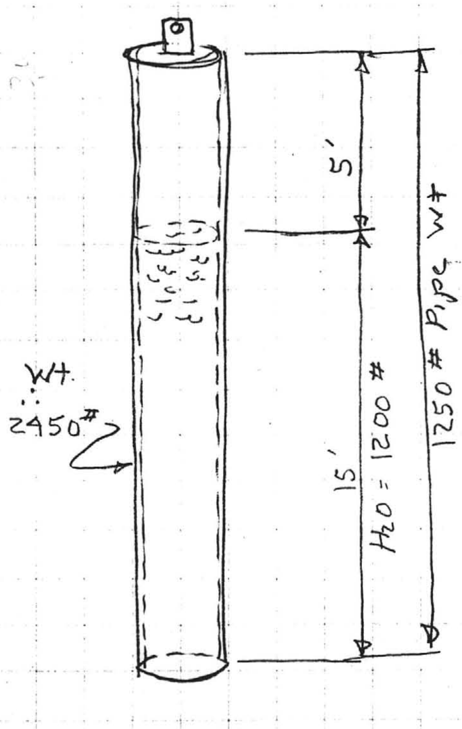
$$\begin{array}{r} 6.409' \\ 4.000 \\ \hline 2 \overline{)\,2.409} \\ 1.2' \end{array} = 1'\text{-}2\tfrac{1}{2}$$

VVX ~ Jerome

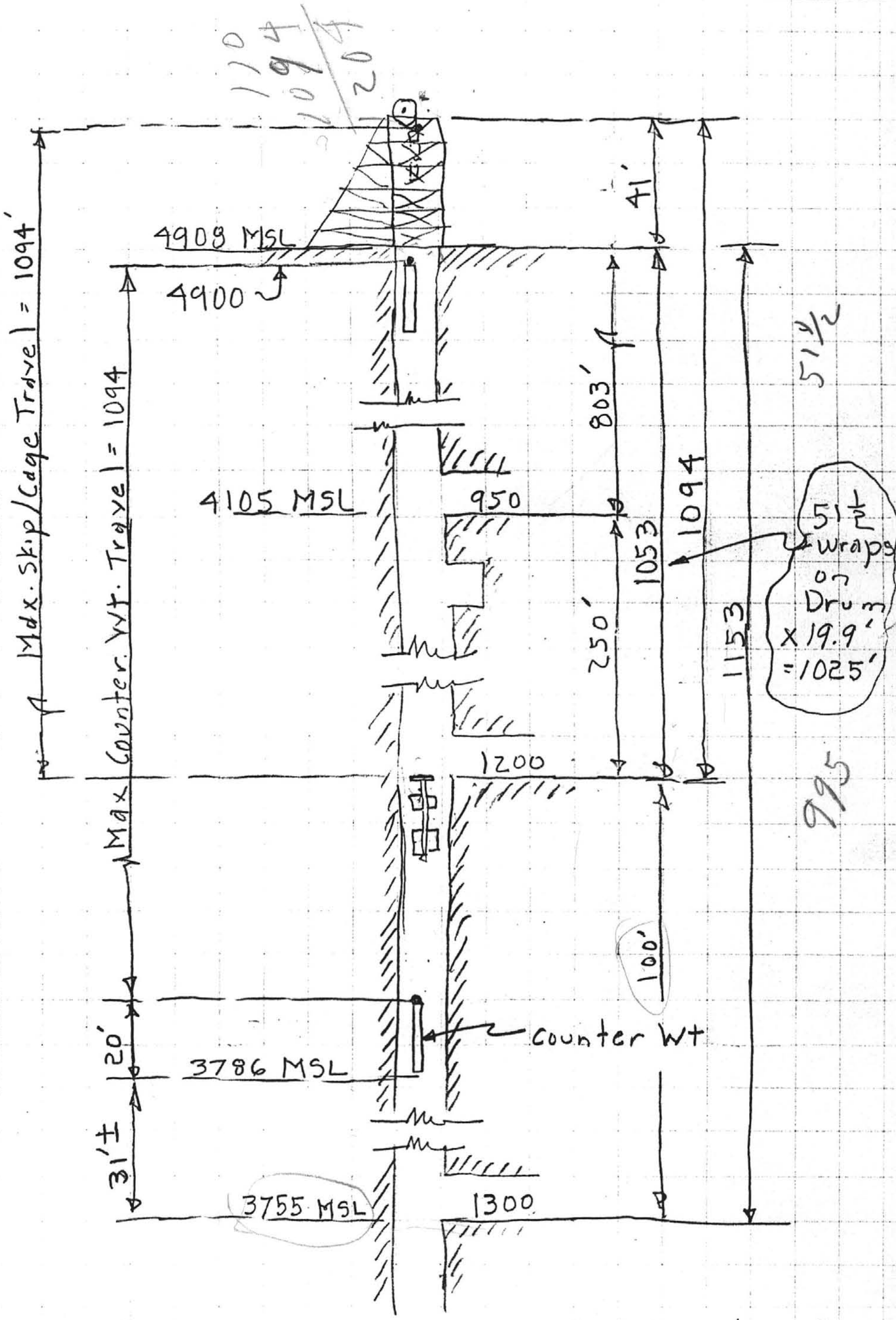
6/8/87



Make 2250#



16" ϕ Std. Pipe



PIPE & WT. DATA

Size	Schedule	Wt./Ft.	Area	ft^3/ft	Mat. Wt./Ft Pipe Wt/ft^3	Wt/Ft Pipe
12 - 12.0 I.D.	S	49.56	113.10 □"	0.785	100	80
→ 14 - 13¼"	S	54.57	137.89	0.958	do	90
16 - 15¼"	S	62.58	182.65	1.268	do	125
18 - 17¼"	S	70.59	233.71	1.623	do	160
20 - 19¼"	S	78.60	291.04	2.021	do	200

UVX ~ JEROME

6/8/87

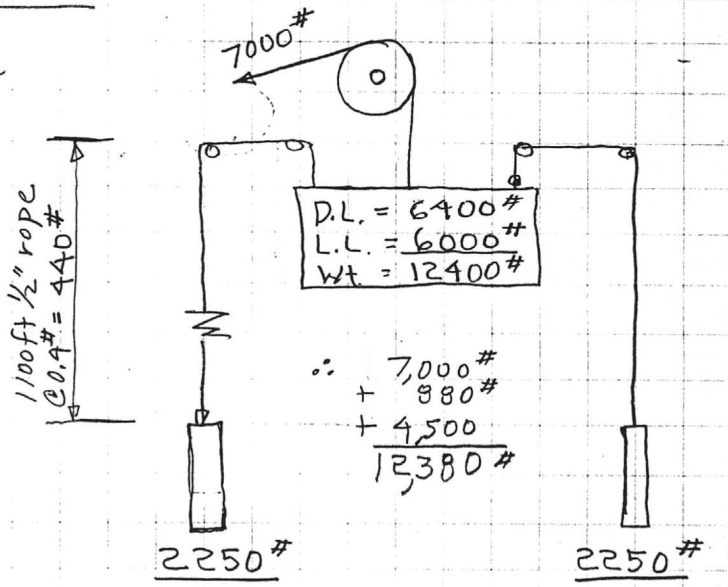
- Skip/Cage Wts.
  - Actual Wt. of old cage = 2,680#
  - " " " " skip = 3,460#

  Total Skip & Cage = 6,140#

* Loads with skip at surface

- Total wt. req'd of counter wts. with skip at surface
- Actual wt. of counter wts = 12,400 − 7,000
  = 5,400# ÷ 2 = 2,700#

  Less rope @ 450
  = 2250#

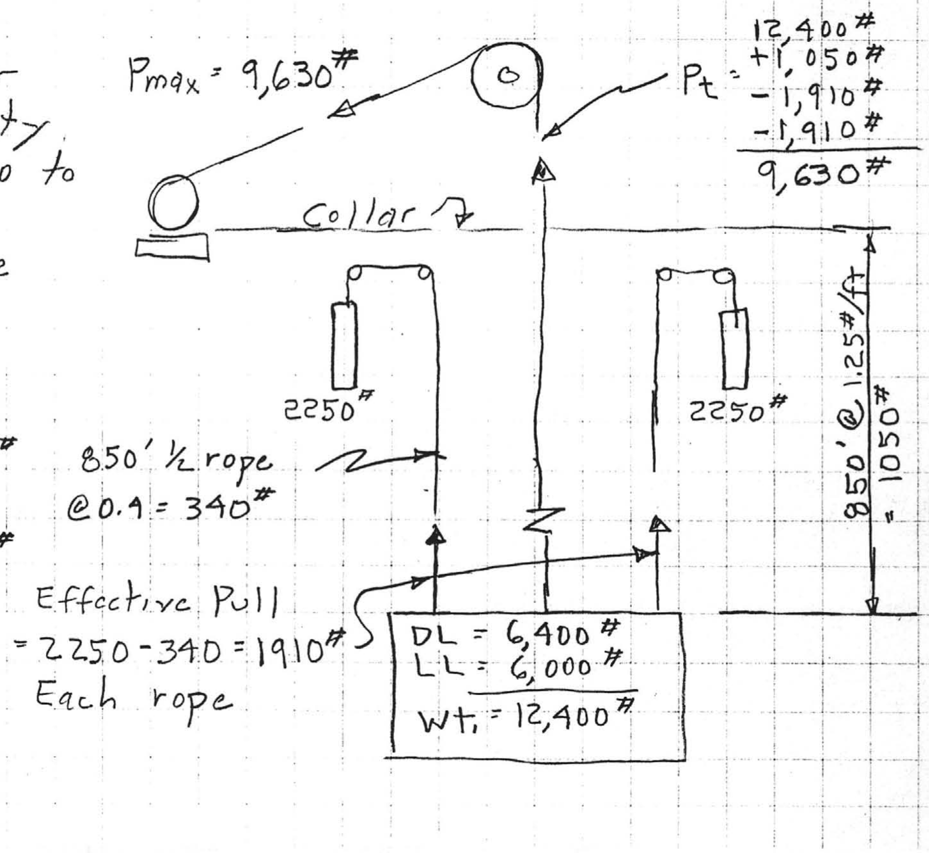


* Loaded Skip @ 1,000 Level

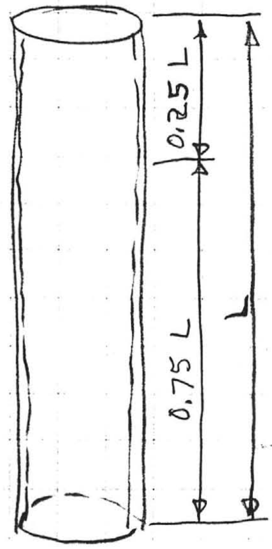
Req'd Saftey Factor
= Hoist rope safety factors for 1,000 to 2,000 ft. heigths
= 6:1 for new rope
= 5:1 for removal

∴ New rope req'd
= 6 × 9,630 = 57,780#

Old rope
= 5 × 9,630 = 48,150#



✓ Counter Weight Data



Size	Wt/ft	ft^3/ft	Wt. Mat.	Length	Wt. @ ¾L	Wt. Pipe	Tot. Wt
12	49.56	0.785	100 #/ft^3	20	1180	990#	2170
16	62.58	1.268	62.4 #/ft^3	20	1190	1250#	2440

A. F. Budge (Mining) Ltd.
UVX
Engineering Summary Sheet
December 1987

902 DDS DRILL HOLE DATA

DDH	Collar Data			Bearing	Inclin.
	Northing	Easting	Elevation		
902-1	11383.64	7208.58	4189.82	S14°30'E	+42°
902-2	11383.18	7205.89	4186.93	S12°00'W	+14°
902-3	11384.46	7202.43	4188.02	S41°30'W	+25°
902-4	11383.65	7208.86	4186.80	S11°30'E	+14°
902-5	11386.60	7201.77	4187.75	S63°30'W	+25°
902-6	11386.49	7202.00	4190.27	S65°00'W	+45°
902-7	11385.93	7203.38	4190.50	S43°00'W	+47°

Notes:

1. The approximate center of rig was coordinates 11390N,7207E.
2. Bearing angles are rounded to the nearest ½ degree.
3. Inclinations are reported to the nearest degree.

A. F. Budge (Mining) Ltd.
UVX
Engineering Summary Sheet
December 1987

902 DDS DRILL HOLE DATA

DDH	Collar Data			Bearing	Inclin.
	Northing	Easting	Elevation		
902-1	11383.64	7208.58	4189.82	S14°30'E	+42°
902-2	11383.18	7205.89	4186.93	S12°00'W	+14°
902-3	11384.46	7202.43	4188.02	S41°30'W	+25°
902-4	11383.65	7208.86	4186.80	S11°30'E	+14°
902-5	11386.60	7201.77	4187.75	S63°30'W	+25°
902-6	11386.49	7202.00	4190.27	S65°00'W	+45°
902-7	11385.93	7203.38	4190.50	S43°00'W	+47°

Notes:

1. The approximate center of rig was coordinates 11390N,7207E.
2. Bearing angles are rounded to the nearest ½ degree.
3. Inclinations are reported to the nearest degree.

A. F. Budge (Mining) Ltd.
UVX
Engineering Summary Sheet
December 24, 1987

901 DDS DRILL HOLE DATA

DDH	Collar Data			Bearing	Inclin.
	Northing	Easting	Elevation		
901-1	11690.0	7753.9	4182.7	S41.5°W	+11°
901-2	11689.4	7753.4	4179.6	S41.5°W	-20°
901-3	11691.3	7750.9	4184.8	S62.5°W	+18°

A. F. Budge (Mining) Ltd.
UVX
Engineering Summary Sheet
December 24, 1987

809 & 806 DDS DRILL HOLE DATA

DDH	Collar Data			Bearing	Inclin.
	Northing	Easting	Elevation		
809-1	11785.4	6909.3	4328.0	S22°W	+25°
809-2	11785.0	6911.9	4328.1	Due South	+27°
809-3	11784.5	6913.1	4325.2	S14°E	-5°
809-4	11786.5	6907.8	4327.7	S40°W	+25°
809-5	11784.6	6911.6	4327.1	S1.5°W	+15°
809-6	11786.3	6908.3	4327.8	S33°W	+25°
809-7	11786.2	6907.5	4327.2	S40.5°W	+18°
809-8	11785.5	6911.8	4329.0	S1.5°W	+35°
809-9	11785.1	6916.8	4325.2	S36°E	-5°
806-1	11899.0	7331.9	4332.4	S31°W	-4°

A. F. Budge (Mining) Ltd.
UVX
Engineering Summary Sheet
December 24, 1987

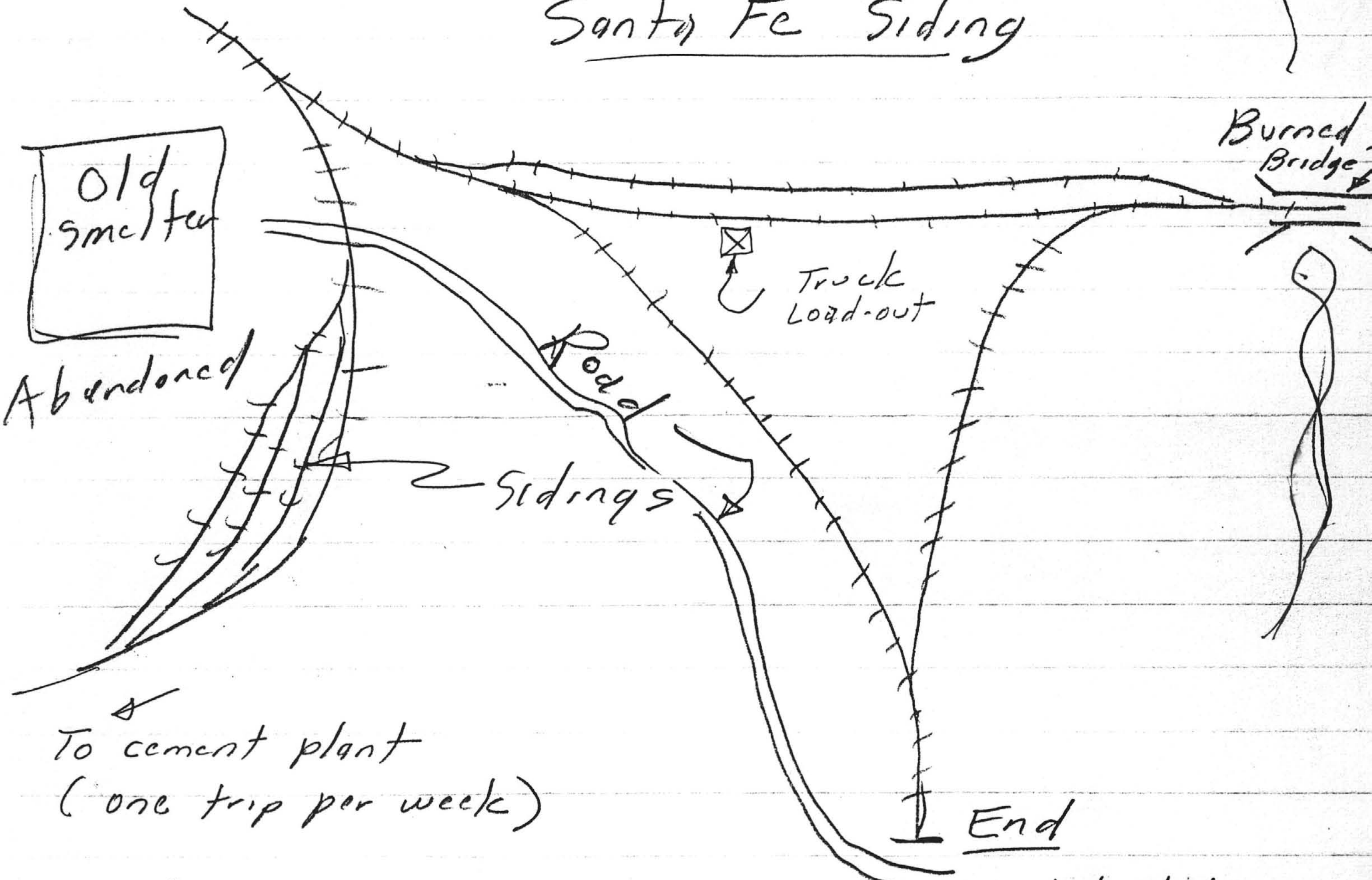
MORGAN DDS DRILL HOLE DATA

DDH	Collar Data			Bearing	Inclin.
	Northing	Easting	Elevation		
M-1	11549.7	7069.6	4189.7	S47.5°W	+45°
M-2	11550.6	7070.6	4190.8	S49°W	+60°
M-3	11547.2	7076.3	4188.4	S15°E	+20°
M-4	11548.8	7075.6	4190.6	S16°E	+50°
M-5	11547.3	7076.1	4187.3	S20.5°E	+11°
M-6	11548.9	7076.2	4189.0	S22°E	+28°
M-7	11547.7	7069.5	4188.3	S46.5°W	+25°
M-8	11547.1	7073.2	4189.7	S17.5°W	+43°
M-9	11545.5	7074.2	4189.5	S6°W	+22°
M-10	11549.6	7077.1	4188.1	S36°E	+22°
M-11	11549.1	7077.2	4188.1	S23.5°E	+25°

Notes: Station Control Point Coordinates: 11555.83N, 7075.47E

Rail Load-Out Siding @ Clarksdale

Santa Fe Siding



Pete said call a guy at Santa Fe, Windslow AZ
named Bill Anderson Ph. 1-~~[illegible]~~
289-7275

Shoemaker
251 3247
Call Mr. Galloway, Phx 251-3264
2920 N. 24th Ave Suite 24, Phx 85015
Galloway head of "Industrial Dept." Santa Fe
Note: Talk of abandoing R.R. to Clarksdale
Cement plant bought by Pima Indians?
Could cause problem to Santa Fe. RR
Leo Kirscht 842 5441

INDEPENDENT
MINING CONSULTANTS, INC.

2708 N. COUNTRY CLUB
TUCSON, ARIZONA 85716
Telephone (602) 323-0868

May 11, 1987

A.J. Fernandez, Senior Mining Engineer
A.F. Budge (Mining) Limited
7340 E. Shoeman Lane, Suite 111 "B" (E)
Scottsdale, Arizona 85251

DMEA LTD.
MAY 13 1987
RECEIVED

Dear Joe,

Attached are our initial thoughts for the potential mining methods which might be employed at the UVX mine. Also attached is a development plan for the 950 level exploration drift. The development plan is based on accessing the drill station locations laid out by Don White. We welcome any comments or suggestions from you on this plan.

At your request, we have also included some discussion of what additional engineering and work is required prior to ore mining. Since the final product(s) will be influenced by the mining method employed, we anticipate a close working relationship between all members of your project team. The time and work proposal is an initial stab at outlining the work required. This is open for review and modification. IMC bills monthly on a time and expenses basis. We consider a day is a day with no additional charge for time over 8 hours. Fraction days are billed based on an 8 hour day. We pass through any expenses with no mark up. Attached is a copy of our current billing rates which will remain constant through our involvement with your project. We strive to work closely with our clients to facilitate communication and maximize the use of everyone's expertise.

IMC is currently available at any level of involvement you would like. The current exploration work will take 3 to 4 months during which I see our involvement as only occasional. Once the additional data is available, we can proceed with the mine engineering work. This work will take about two months. This places the final go decision around six months from now (November - December 1987).

I want to thank you and your staff, as well as, Carole and Don for making Jack's and my visit to Jerome very informative. You have an interesting project and we look forward to working with you on it. Please do not hesitate to call if you have any questions or comments.

Regards,

Herbert E Welhener

Herbert E. Welhener
Principal Mining Engineer

1.0 DEVELOPMENT AND EXPLORATION DRIFT - 950 LEVEL

A preliminary layout for the mine development work recommended to provide access to the proposed 902 and 906 diamond drill stations (D.D.S.) is shown on Figure 1-1. Approximately 750 linear feet of 6 ft wide x 7 ft high drift and turnouts are required by the plan plus the excavation of the two drill stations.

The recommended layout allows for satisfactory access to the 902 and 906 drill station. It provides strategically located turnouts that can be used for later orebody delineation drilling and/or stope preparation work, and it facilitates the eventual completion of a rail haulage loop that will be needed for mine production purposes. A description of the layout follows:

1. From a point in the Morgan Cross Cut (X-Cut) approximately 30 ft back from the present face of the Morgan D.D.S, drive an access drift to the south towards the wider portion of the M-3 ore zone; then turn to the southeast and proceed to the location of the 902 D.D.S. During the course of driving the access drift, excavate turnouts #1 and #2. Turnout #1 will provide easy, immediate access to the M-3 zone for bulk sampling pruposes.

2. Excavate the 902 D.D.S. on the southwest side of the access drift in order that diamond drilling and drift driving can be carried out simultaneously.

3. Continue the access drift to the southeast, excavate turnouts #3 and #4, and drive 40 ft past the location of the 906 D.D.S. to intersect 906 X-Cut. The portion of the access drift between turnout #3 and the 906 X-Cut will also intersect X-Cuts 902-W and 990. These three old X-Cuts may allow entry to presently inaccessible areas of the 950 level.

4. Excavate the 906 D.D.S.

The access drift should be driven from the Morgan X-Cut on as flat a grade as practical in order that it's inersections with the old X-Cuts will be at similar elevations.

Turnout #3 can be used as a drill station from which to explore the area immediately south of the Gold Stope, or it may be needed for completion of the haulage loop from the 901-S X-Cut.

The estimated cost for the 750 feet of drifting and turnouts plus the excavation of two diamond drill stations is $210,000. Approximately three months will be required to complete the work if operations are carried out on a two shift per day, five days per week working schedule.

2.0 POTENTIAL MINING METHODS

The selection of suitable mining methods for U.V.X. orebodies is severely restricted by the incompetent ground conditions within the ore zones, by the limited size of the Edith Shaft compartments, and by the prohibition on any additional surface subsidence. Only two stoping methods can be reasonably expected to meet these limitations. They are the conventional (non-mechanized) cut and fill system or the square-set mining system. Unfortunately, both of these options are relatively high-cost, labor-intensive stoping methods. Both methods require backfilling of the mined out areas with waste material to prevent surface subsidence, and both require that alternate stoping areas be made available so that mine production does not suffer during the backfilling cycle.

It is anticipated that cut and fill mining may possibly be employed in certain portions of the lower grade ore zones where ground conditions are somewhat reasonable, but the high grade, siliceous-grit ores will,in all probability, require square set stoping for adequate ground control.

Presently, conventional cut and fill mining is extensively used in the Idaho silver belt mines where poor ground conditions are commonplace. Mining costs range from $40.00 to $60.00 per ton of ore with the smaller daily tonnage mines experiencing the higher costs. U.V.X. cut and fill mining will probably be in the same cost range as the smaller Idaho producers.

Square set mining with cemented backfill was commonly used in heavy ground by Phelps Dodge at Bisbee, Arizona until the mines were closed in 1975. P.D.'s costs, escalated to 1987 dollars, would be approximately $90-$100/ton.

Relatively small pieces of mining equipment (locomotives, cars, mucking machines, slushers, etc.) must be used at U.V.X. because of the limited dimensions of the Edith Shaft. The mining equipment is similar for either conventional cut and fill or square set mining so both systems may be used concurrently in different areas of the mine. Square set mining does, however, require a large amount of timber. Consequently, planning should include suitable surface location for a timber framing shop and a timber storage yard.

3.0 ENGINEERING REQUIRED FOR MINING METHOD SELECTION

The selection of the mining method will be dependent in part on the destination of the mined material. A trade off should be evaluated between mining all the gold bearing ore and processing it for the gold only, or mining the ore selectively and producing multiple products. This work should be in enough detail through final sale of product or metal to determine which approach is more economic.

The drilling data will be entered into the IMC computer. This will provide a data base for reserve estimation and the potential of building a small reserve model. The data base will provide rapid access to the drilling information for plotting sections and levels at any orientation, and estimating reserves for various mining approaches or schedules.

Once a mining approach is selected, an operational mine plan can be developed. This mine plan will include development and stoping plans for the initial ore pods to be mined. A timetable for development, ore production, and continued exploration drilling will be provided. This continued exploration will be required due to the irregularity in size and shape of the ore zones. These zones need to be well defined prior to development for mining. A mining schedule will be prepared which will assure continuous ore production. Based on this mining schedule, manpower and equipment requirement will be determined and capital and operating costs estimated. Table 1 shows a time and cost estimate to complete this work. The calendar time frame for the engineering work will be around two months.

The mining selection engineering will be done after the diamond drilling is completed from the new exploration drift on the 950 level. This should provide a better estimate of the ore pods' shape and size. Once the inplace reserves are determined, the recoverable reserves by each mining method can be estimated. The current knowledge of the ore geometry from the M-3 pod is a start but is not extensive enough to base a complete development plan on. It is believed that the ore character and geometry is similar to that in the Gold stope, but with the closer proximity to the Verde Fault in the current target area, conditions could be different.

The time frame to complete the new exploration drift and drilling will be 3 to 4 months. Other activities can be going on while this work is being completed. Items related to mining include:

1. determination of the inplace density for each of the three major ore types (the siliceous grit, the more competent chert and the ironstone)

2. extend the drift from the turnout #1 into the M-3 ore pod

3. establish a surface subsidence monitoring network

4. collect samples for strength testing in diorite and the various ore types.

The determination of the inplace density is critical to the reserve estimation. During the site visit on May 7, core from the M-3 zone was observed with Don White. Holding pieces of core from the three ore types demonstrated a large difference between the siliceous grit (quite light, perhaps 15 to 16 cu ft/ton) and the ironstone (heavy by comparison, perhaps 10 to 11 cu ft/ton). The higher grade zones are in the siliceous grit and the density of these will impact the total ounces markedly.

The extension of the drift into the M-3 ore zone and taking some bulk samples will provide material for metallurgical testing. This can be for both gold recoveries from the various ore types and the flux qualities for potential smelter sales. As well, it will provide some experience on the ability of segragating the ore types during mining.

Mining in the Verde area cannot impart any additional surface subsidence. It is known that some subsidence is currently happening. This subsidence should be documented and a monitoring program established prior to any ore mining. This will establish the current subsidence picture and provide the data to show that the new mining has or has not impacted this subsidence.

Table 1

UVX Mining Plan

Feasibility Study and Operational Mine Plan
Time and Cost Estimate

Establish Drill Hole Reserve Base		
Mine Engineer	8 Days @ $400/Day	$ 3,200.00
Jr. Engineer	4 Days @ $250/Day	1,000.00
Computer	4 Hrs. @ $200/Hour	800.00
		5,000.00
Selection of Final Mining Method		
Mine Engineer	10 Days @ $400/Day	4,000.00
Operational Mine Plan and Schedule on Final Mining Method		
Mine Engineer	6 Days @ $400/Day	2,400.00
Jr. Engineer	4 Days @ $250/Day	1,000.00
Computer	2 Hrs. @ $200/Hour	400.00
		3,800.00
Equipmemt amd Manpower Requirements		
Mine Engineer	5 Days @ $400/Day	2,000.00
Capital and Operation Cost Estimate		
Mine Engineer	5 Days @ $400/Day	2,000.00
Report to document work		
Mine Engineer	5 Days @ $400/Day	2,000.00
Draftsman	4 Days @ $150/Day	600.00
Clerical	5 Days @ $120/Day	600.00
		3,200.00
Supplies, Travel, Reproduction, etc.		250.00
Total Estimate		$20,250.00

Independent Mining Consultants

Billing Rates

Principal Mining Engineer	$400 per day
Mike Hester John Marek Herb Welhener	
Court Appearances for Above Personnel	$600 per day
Junior Mining Engineer	$250 per day
Yvette Riddle	
Associates and Consultants	
Robert Butler	$300 per day
Ed Jucevic	$400 per day
Jack Riddle	$400 per day
Robert Shoemaker	$800 per day
Robert Thurmond	$500 per day
Drafting	$150 per day
Clerical	$120 per day
HP-9000 Computer	$200 per Computational Hr.
Floating Cone Pit Bounding computer run (Not to Exceed)	
First Multi-Slope Run	$3500
Subsequent Multi-Slope Runs	$3000
First Single Slope Run	$2500
Expenses	At Cost
Reproduction, Photocopies, Telephone, Travel	

January 1, 1987

INDEPENDENT
MINING CONSULTANTS, INC.

INTER-OFFICE

TO: A. F. Budge Mining Ltd. DATE: March 12, 1987

FROM: H. G. King

SUBJECT: Planning for Future Mine Development at UVX Mine

LOCATION: Jerome, AZ MEETING DATE: March 11, 1987

PRESENT: Carol O'Brien, Pete Flores, Joe Fernandez, Howard King, and Don White (part-time)

° Primary subject of discussion was to evaluate the need to design, construct, and install a skip at the Edith shaft of the UVX mine provided continuing mine development becomes economically feasible.

° To determine a basis for skip design, future mine operational requirements were assumed to be as follows:

- Skip would be designed to take the maximum load possible with the existing shaft size, headframe, and hoist installation. (Hopefully this would allow the mine to produce 100 tons of rock from underground per shift.)

- Mine would operate two shift per day.

- Main skip loading station would be designed and built for loading from the 950 level since the great majority of production will come from that level.

- Loading at 1100 level could be done at a later date but with slightly less skip capacity and less sophisticated loading pocket.

- Future surface ore processing plant would be operated on the day shift only.

- Primary crushing could take place underground at the 950 level which would have some effect on the skip design.

- The man cage will likely be suspended under the skip for operational purposes.

° Mine operations will probably need the skip in the near future if further development of the mine proves feasible since waste disposal areas are no longer readily available underground.

° Joe Fernandez will furnish the maximum unit weights and hardness of rock types that will be passed through the system in order to determine crushing requirements and optimum skip size.

- Current thinking is that a surface processing plant would be built to pass R.O.M. waste through to a surface dump site and to reduce ore to ¼-inch by zero and 2-inch by ¼-inch sizes.

- Due to environmental considerations, the surface processing plant would be operated on the day shift only.

- A search will be made for a used skip, and, if unavailable, both in-house designs and purchase from a skip manufacturing company will be economically evaluated.

- Until future extensive mine production proves feasible, the only new materials handling plant facility to enter into advanced design at this time will be the skip and necessary modifications to the headframe and loading station to accommodate the skip.

H. G. King

H. G. King

copy: Carol O'Brien
Pete Flores
Joe Fernandez

Carde

MEMORANDUM

TO: D.C. White and C.A. O'Brien

FROM: R.W. Hodder

DATE: April 30, 1988

SUBJECT: Memo by P.A. Handverger, April 20, 1988
re: Gold potential of property peripheral to current UVX Project

I am not inclined to rush into any of the 3 deep, primary sulphide targets proposed by Paul in his memo of April 20, 1988. Drill holes AV-27, AV-10 and B-1 were drilled to explore for primary pyritic base metal sulphides adjacent to Cleopatra quartz porphyry and at best encountered gold abundances typical of this target. Our work at UVX to date suggests greatest gold concentration may be:

1, At least in part secondary, or supergene, and about 200 feet below the Cambrian/Precambrian unconformity;
2, In repeatedly broken rock of the upper sequence intruded by diorite and in the hanging wall of the Verde fault zone;
3, Characterized by intense silicification overprinted by secondary copper minerals, iron oxide minerals, and gold in that order with decreasing elevation;
4, Downhill from the United Verde which may have been a source from which surface waters transported silica, copper, iron and gold to a site of deposition focused by the intersection of the Verde fault zone and cross structures such as the Florencia fault. Bitter Creek appears to continue this transportation and deposition of these elements to this day and offers an instructive modern analogue.

Hence, there are better possibilities of gold concentrations comparable to the UVX nearer surface, along the Verde fault zone at cross structures and near the Cambrian/Precambrian unconformity. These can be sought initially through 3 days of traversing looking for dense quartz, secondary copper minerals, iron oxides in the aforementioned structural sites. Existing maps can help orient this traversing.

The target-type Paul proposes, one of gold in primary sulphides in the base metal massive sulphide environment is much more attractive logistically and in economic potential at the United Verde where it is relatively near surface, amidst considerable data acquired in previous mining operations and surface mapping and accessible through short drill holes.

In brief, I would take this opportunity to look at the core from Paul's three holes, not so much for the primary target but for secondary mineralization in the upper sequence and I would persist in approaching P.D. for a look at the United Verde. I also recommend three days of field traverses along the Verde Fault for UVX-type material which might have been considered in the past only for an upper secondary copper enrichment. In this manner we can use data developed at UVX and gain data useful in further understanding the UVX metal distribution.

Carole

Don White
521 East Willis St.
Prescott, AZ 86301

Date: April 25, 1988

Robert Crook / Jim Weatherby
Iron King Assay, Inc.
P.O. Box 56
Humboldt, AZ 86329
(632-7410)

778-3140

UVX Batch # 125

Hello Bob + Jim + Kati;

Accompanying are eight (8) samples for one assay ton gold and silver fire assay with AA following as appropriate. The samples are numbered:

1 906 -101
2 906 -102
3 906 -103
4 -104
5 -105
6 -106
7 -107
8 906 -108

Select lithologies from 906-903 raise debris

Please save all pulps + rejects for my pickup.
Please send a copy of the results + billing to Carole (below)

c.c. Carole A. O'Brien
A.F. Budge (Mining) Ltd.
7340 East Shoeman Ln.
Suite 111-B-(E)
Scottsdale, AZ 85251

Thanks,
Don White
Don White
Geologist, CPG.

R.W.H. says Mae Gustin also at U. of Cincinnati

Conde

University of Cincinnati

Department of Geology

547 Geology/Physics Building
Cincinnati, Ohio 45221-0013

April 11, 1988

Mr. Don C. White
521 East Willis St.
Prescott, AZ 86301

Dear Don:

Thanks so much for taking the time to present an impromptu talk about your work on the UVX's gold deposits. It was most informative and opened my eyes about the gold-bearing siliceous ore bodies. The students also enjoyed your talk and seeing an exploration/ development project in progress.

I am very interested in working with you on the identification of mineral phases in the different zones that you have described. I have spoken with Tiebing Liu, a doctoral student who was on the field trip in March. He is finishing his dissertation, but has some time to do a side project. So, he will conduct most of the analyses of core samples that you send us. I would appreciate receiving enough of each sample for: (1) a thin section (where possible), (2) X-ray diffraction (approximately 5cc) and (3) possible SEM work. I suggest approaching the identification problem by running X-ray powder diffraction on each sample and looking at polished thin sections of the samples. A small subset would be chosen for scanning Electron Microprobe analysis, in order to attempt to determine the different iron oxide/hydroxide phases. (You mentioned that these are diagnostic, with tan-yellow being associated with precious metals. Am I correct in recalling that some deep red iron oxides are <u>not</u> associated with precious metals?) The SEM is in another department, and we are charged for this. Thus, the bulk of analytical costs will pertain to SEM work. However, this will likely produce the best information for you. I don't have the cost of SEM work, yet, but I'll call with that information. Do you plan to send samples from at least <u>two</u> drillholes? I would suggest at least 2 drillholes and/or 2 samples <u>within</u> each alteration zone so we can sense the variability of mineralogy. If you can send them copies of the drillcore, the logs would be helpful for placing these samples in context. What number of samples are we talking about? My sense was 20-30, but that may not be correct.

I've been thinking about your description of these deposits, based on my sketchy notes. I am trying to envision these quartz-rich mounds forming by repeated brecciation, and need clarification on a few items:

1. Does the brecciation extend well beyond the ≤0.1 opt Au halo atop

these mounds, or is this the point where hematitic chert appears?

2. Is there a <u>true</u> ferruginous <u>chert</u> overlying the breccias, or is it ferruginous quartz that is also brecciated?

3. If there is overlying ferruginous chert, is it ever brecciated? (i.e. did the repeated pulses of fracturing continue <u>after</u> chemical sediments were being deposited?)

These help establish the timing of mineralization and whether mineralization required brecciation (versus concentration to high grades).

I am also curious about your comments about the base metals perhaps <u>not</u> being deposited as sulfides. This sounds very interesting, as it may reflect the sulfur-oxygen conditions during deposition. We can look for any possible sulfide replacement textures, particularly with SEM. However, I don't know whether this is part of some other project. (You mentioned one at U. of Arizona and one of Western Ontario.) I also realize that this is not your primary concern at the moment.

Enclosed are pertinent portions of a paper by Huston and Large which you may find interesting. They have also submitted an article to Economic Geology, which has been or is about to be printed. Ross Large sent me this paper. It would be interesting to compare the UVX with his model of gold deposition.

I hope the paper is useful to you. I look forward to receiving samples and answers to my questions. If you have any questions, please call me at (513) 475-5195. I am in and out of my office quite a bit; if you don't reach me, leave a message at (513) 475-3732 and I will call back. I would like to know how many samples you want analyzed. Tiebing can start work this spring, but will probably continue the project into the summer, under my supervision. I hope that this timing fits your schedule, as I am excited about the project.

Sincerely,

Holly Huyck

Holly Huyck
Assistant Professor

P.S. Results of this work will be considered proprietary and kept confidential. I appreciate your openness in discussing the UVX deposits with our group.

CAO'B

U. S. Geological Survey
Branch of Geochemistry
Box 25Ø46, MS 973
Denver, CO 8Ø225

April 21, 1988

Mr. Don White
521 East Willis Street
Prescott, AZ 86301

Dear Don,

Thanks for sending the package of new information and the excellent series of photos of face exposures underground. I'm relieved that my comments of March 18 do not clash terribly with your more extensive observations on the complex gold-silica zones--I was concerned that my initial observations might offend you.

I've rechecked the sections with an eye on the gold assays; the first observations were not keyed to gold content in an effort to be unbiased. The gold rich samples, particularly 906-5 and 902-4 to 902-6 have an abundance of 50-100 micron quartz that is complexly intergrown with much junky iron oxides and also with cubic or rectangular voids that are partly filled by silica. The very high grade site 906-12 has a particularly distinctive very fine grained chalcedonic (10-20 micron) intergrowth of quartz. There are many instances of delicate growth zones in the quartz, and many euhedral crystals--features commonly observed in quartz veins or replacements of shallow level Tertiary systems and termed "epithermal" by most observers. Two photos of 906-5 show some of the features (I ran out of film and did not shoot the many other examples I refer to).

There are many aspects of the siliceous rocks that need much more study. Some chief observations or questions that come to mind include the following. 1. The fine quartz is unstrained, thus is post-metamorphic? 2. There may be some clasts of this kind of quartz, indicating tectonic breakage in the Tertiary?--there probably is information here to help work out limits on some ages relative to faulting. 3. The "epithermal" unstrained quartz as in 906-5 has intergrown hematite--the iron oxides in this situation are not surface coatings--and such probable co-precipitation could place constraints on several chemical conditions and permit interpretation of fluid character such as oxidizing hypogene, or could such coprecipitation come from some unusual kind of deep supergene process? (not my inclination, but food for thought?). 4. The rectangular ghosts are obvious in form, but tough to identify. Chemistry might show traces of something like Ba to help the investigation, and possibly closer scrutiny would reveal partially preserved relicts. 5. Additional sampling would add to the story--a thin section slice is a real small sample and may not reliably represent an assay or field observation of structure and lithology. 6. Gold minerals presumably can be found in the samples, but I've not attempted that. It might be possible to place the gold grains in a mineralizing sequence, or the patterns might be ambiguous. I would not expect to be able to track anything to a pre-metamorphic protolith because these rocks have been through some many stages of metamorphism and alteration.

I want to emphasize that all my observations are preliminary in nature and in need of confirmation by additional work. But I offer them for your consideration because you have been so supportive of research on the gold-silica zones. At this time I'm running in circles trying to complete some old projects and going to meetings and other busy stuff before I go to Nevada to start a new program of geochemical studies. We are gearing up for a new megaproject of geologic-geochemical-geophysical studies attempting to develop methods for assessing mineral potential under post-mineral cover, ie exploration methods for concealed deposits. That is a very tough problem, as I'm sure you would say, and possibly I can provide some ground truth to keep things in line; anyway, it is a political mandate and that is where the action must be!

I'll be back in touch with you when I can, but for the next few months there will be not time for a trip to Jerome, although I appreciate your cordial invitation.

Best regards,

Tom

J. Thomas Nash
Research Geologist

LASZLO DUDAS

MINERALOGIST

MINERALOGICAL CONSULTATION
ORE MICROSCOPY
PETROGRAPHY
ECONOMIC GEOLOGY
(ANALYSIS & INTERPRETATION)

PH: 602
795-4251

4737 E. ADAMS
TUCSON, AZ. 85712

March 9, 1989

Mr. Dale Allen
Chief Metallurgist
A.F.Budge Mining, Ltd.
4301 North 75th St. Suite 101
Scottsdale, AZ 85251-3504

Dear Dale:

Enclosed please find the mineralogical report on the gold ore from United Verde Extension Mine.

The microscopic observation of the received hand samples at 30x magnification did not reveal any visible gold, silver or sulfide minerals thus it was assumed that any economically valuable mineral(s) occurs only in very fine size. For this reason it was decided to make two thin and two polished sections of all samples, including the hand, the flotation test samples and their screen and gravity products.

No gold was found in the polished sections of the chert. The consistency of the "grit" prevented the preparation of a coherent thin and polished section, thus it had to be crushed and ground to make thin and polished sections from "as is" screen and gravity fractions. The same procedure was applied to the flotation test feed. The rougher tailing - leach residue was finely ground so it did not need any extra preparation.

The microscopic examination showed that the gold occurs in very fine, free and locked particles. The silver is represented by cerargyrite, a silver chloride. Their recovery is hindered not only by the fine size, but also by the overwhelming hydrous iron impregnation of the devitrified glass (which is one of the major transparent component in the ore). The fine size gold imbedded in quartz needs a very fine grind which would require a slime flotation circuit in the mill to avoid excessive loss. In addition to this, very high amount of cyanide is needed to overcome the deleterious effect of the readily soluable hydrous iron oxides, thus enough free cyanide must be secured to recover all the gold and silver (all the thin and polished sections,-except that of the "chert's" - exhibit strong reddish-brown color indicating high iron content).

All in all the characteristics of the ore raises the question of economy whether or not would it be profitable to recover the gold and silver by flotation and leaching.

MINERALOGICAL REPORT.

To: Mr. Dale Allen
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From: Laszlo Dudas
Mineralogist
4737 E Adams St.
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Subject: Mineralogical Investigation of a Gold Ore and its Metallurgical Test Products from the United Verde Extension Mine, Jerome, Yavapai County, Arizona.

Purpose: To determine the mineral composition of the samples in particular respect to their gold and silver mineralization, their locking properties and to give information about any deleterious minerals adversely affecting the noble metal recovery.

Samples: Two hand samples, each weighing approximately 500 grams and two flotation products weighing approximately 150 grams were received from Mr. John A. McKenney and Ms. Carole A. O'Brien on February 7, 1989 as follows:

1. High Grade "Chert", Sample No. 809,
2. High Grade "Grit" Sample No. 902,
3. Flotation Feed (minus 35 mesh), Composite No.1, Test No. 9.
4. Flotation Rougher Tailings-Leach Residue, Composite No. 1, Test No. 9.

The above received samples, due to their variable consistency, required different processes of preparation for mineralogical study. The two hand samples ("chert" and "grit") were first observed under a binocular stereoscopic microscope at 30x magnification to gain information about the macroscopic distribution of their mineral components.

The "chert" is a dense, hard quartz showing very small dark areas. Two thin and two polished sections were prepared from these preferred areas.

The "grit" consists of angular, subangular quartz grains losely bound with coffee brown fine, sandy (tuffaceous?) matrix. The sample is very crumbly thus it failed to render coherent thin and polished sections. Because of this, pieces of grit weighing 100 grams were subjected to crushing by a stainless steel rolling pin and two thin sections were made from this crushed sample. This rough, crushed sample was screened on a 100 mesh sieve. The resulting screen fractions were weighed and two polished sections were made from each of the plus and minus 100 mesh fractions. Then the plus 100 mesh screen fraction was ground to 100 percent minus 100 mesh and both minus 100 mesh (screen and ground) fractions were thoroughly mixed. This mixed minus 100 mesh fraction represents the total ore in minus 100 mesh size which is in most cases (not in this sample) the practical liberation size of any economically valuable mineral. No sections were made from this mixed total ore. However, a twenty gram sample was cut out of this mixed minus 100 mesh total ore and subjected to a gravity concentration by Haultain Superpanner. The resulting gravity concentrate and tailings were dried and weighed. Two thin and two polished sections were prepared from each of the gravity concentrate and gravity tailings.

The flotation test feed (minus 35 mesh) was first screened on a 100 mesh sieve to avoid severe plucking in polish and thin sections caused by the great difference in size and hardness of the mineral particles. The resulting size fractions were weighed. Two thin and two polished sections were prepared from each of the plus and minus 100 mesh fractions. Then the plus 100 mesh screen fraction was ground 100 percent to minus 100 mesh. The two minus 100 mesh (screen and ground) fractions were thoroughly mixed. The mixed minus 100 mesh represents the total ore in minus 100 mesh size. (The reason for this preparation step was explained in the above description of the preparation of the "grit" sample.) Two thin and two polished sections were made from the mixed total ore. After this 20 grams were cut out of the mixed minus 100 mesh total ore and subjected to gravity separation by a Haultain Superpanner. The obtained gravity fractions were dried and weighed. Two thin and two polished sections were prepared from each of the gravity concentrate and gravity tailings.

The rougher tailings-leach residue was already reground (88 percent to minus 400 mesh) by Dawson Metallurgical Laboratories Inc. Thus two thin and two polished sections were made from the sample in "as is" (as received) condition. Then 20 grams were taken from the sample and were subjected to a gravity separation by a Haultain Superpanner. The resulting gravity concentrate and gravity tailings were dried and weighed. Two thin and two polished sections were prepared from each of the gravity fractions. All the 24 thin and 24 polished sections were thoroughly examined under reflected and transmitted light polarizing microscopes respectively. Results follow.

D I S C U S S I O N .

1. The designations of the two hand samples received for the present mineralogical investigation, "Chert" and "Grit" usually implies to sedimentary minerals or rock formation. The microscopic observation under a stereo bincoular microscope at 30x magnification showed that the "chert" is a dense, hard, moderately fractured quartz for which the name chert, in strict sense, may or may not be aplicable. The "grit", however, under the same magnification appears to be a poorly consolidated, coffee brown, crumbly sandstone. Regarding its consistency no thin section could be made from a coherent piece, thus the microscopic examination of the thin sections of the grit is restricted to crushed and ground "as is" sample and its flotation products. The microscopic examination of the thin sections of both samples ("chert" and "grit") definitely indicates their igneous origin.

2. Since all the received samples (hand and flotation test products) originate from the same deposit their mineral components will be described jointly.

3. TRANSPARENT MINERALS predominate in all the received samples (hand samples and flotation products).

Quartz is the dominant transparent mineral in all samples. It occurs in two types: (1) very fine grained, forming the matrix of the chert, and (2) medium size grains filling fractures. It appears that the "chert" actually represents an early hydrothermal vein quartz which was subjected to pressure and recrystallized to an uniformly fine grained (10-15 micron size) mass. This quartz is heavily fractured and the cracks, fractures and perhaps the preexistent vugs are filled or healed with a younger quartz generation of medium size grains (60 to 150 micron size).

Since no coherent thin section could be made from the "grit" pieces the description of the quartz in this sample is restricted to the crushed and ground grains. The quartz apparently occurs in angular to subangular medium size grains in the "grit". Occasionally a few fine grained quartz particles may be present in the plus 100 mesh screen fraction and the gravity tailings of the flotation test feed.

Most of the quartz is free, only a small amount of it is locked with hematite and hydrous iron oxides (one to eight percent in the head samples and also in the flotation test products, see Tables I to IV.). The quartz carries rutile in small pieces (15 to 50 microns) as locking component. Pyrite is also a frequent locking component in very small sizes (two to 15 microns) in the quartz. Gold occurs in trace amounts (four to 12 microns in average) imbedded in the quartz. On one occasion a 45x27 micron gold particle was seen in a fracture of a quartz grain. The distribution of the gold as locking component in the quartz is very low (one or two in a few polished section).

Devitrified glass is the second most abundant transparent component in the "grit" hand sample and in all the flotation test products (see Tables I and III). It appears that the devitrified glass served as matrix for the losely bound, angular to subangular quartz grains of a volcanic tuff. The devitrified glass consists of minor amount of fine grained equigranular quartz and mostly of feldspar and perhaps clay. Devitrification actually means the beginning of crystallization of the volcanic glass, thus the identification of the types of feldspars and clays is possible only by microprobe analyses. This transformation of the glass to individual components makes the components somewhat porous. For this reason only a small amount of the devitrified glass remains clean (four to 10 volume percent), the larger portion of it is heavily impregnated by hydrous iron oxides (14 to 48 volume percent) hence the coffee brown color of the "grit". All the thin and polished sections show strong reddish-brown color under polarizing microscope (see Tables I and II). The term of impregnation means that the circulating surface water carries considerable amount of iron in solution which soaks the porous, divitrified glass (of the tuff) and precipitates as hydrous iron oxides of the limonite group of minerals in the pores.

For practical purposes all the transparent minerals (e.g. quartz and devitrified glass) are combined under the heading "Transparent Gangue" in the flotation test products. Thus the transparent gangue overwhelmingly predominates in all the samples examined (88 to 99 volume percent, see Tables III and IV).

Malachite is a minor trace mineral in the flotation test products (see Table II). The grain size ranges from 40 to 120 microns.

Chlorite is a subtrace component in the minus 100 mixed total ore.

4. OPAQUE MINERALS are minor components in all the received samples. The discrepancy showing in the distribution of the opaque minerals presented in the tables (between Tables I-II and Tables III-IV) is due to the difference in magnification between transmitted (80x) and reflected light (160x) polarizing microscopes, consequnetly the field of view is twice as large at the lower than at

the higher magnification.

Hematite is the major opaque mineral in the received samples. About half of the hematite is free (one to five volume percent) in the flotation test products, the other half occurs as locking component in the transparent gangue (two to eight volume percent, see Table IV). Most of the hematite is present as locking component in the hand samples of "chert" and "grit" (see Table III). The hematite occurs in small to medium size (40 to 150 microns) round or elongated grains, often forming porous free aggregates or is intergrown with quartz (the aggregate size reaches 180 to 200 microns).

Hydrous Iron Oxides are the second most frequent opaque mineral components in the samples. They are present in trace amounts in the hand samples of the "chert" and "grit" (see Table III.). About half of the hydrous iron oxides (similarly to hematite) occur as free grains (trace to four volume percent), the other half as locking components in the quartz (one to six volume percent, see Table IV). The hydrous iron oxides comprise goethite and mostly the limonite group of minerals. Some or most of the hydrous iron oxides appear as alteration products of a preexistent pyrite. One hydrous iron oxide grain (about 150 microns) carries a native gold flake about 38x45 microns.

Rutile is a frequent minor component in the hand samples of the "chert" and "grit" (one volume percent, see Table III) as well as in the flotation test products (one to three volume percent). It is present in small grains (15-40 microns) exclusively as locking component in the quartz. Because of the fine grind (88 weight percent minus 400 mesh) of the rougher tailings-leach residue no rutile could be detected in the quartz grains.

Pyrite is a frequent trace mineral in the hand samples of the "chert" and "grit" (see Table III). It occurs as locking component of very small size (tiny specs and grains) in the quartz. Pyrite, free and as locking components, is common in the flotation products in trace amounts (two volume percent of free pyrite occurs in the gravity concentrate of the rougher tailings-leach residue, see Table IV). The grain size of the free pyrite varies between 20 and 80 microns, large sizes (80 microns) are rare.

Chalcopyrite was spotted only in the gravity concentrate of the flotation test feed. The one grain measured 30x80 microns.

Native Gold is a moderately frequent component in some samples. No native gold was seen in the polished section of the "chert" . Native gold, however, occurs as locking component in quartz and as free flakes (the size varies between four and 12 microns) in the gravity concentrate and tailings of the ground hand sample of the "grit" (see Table III).

Similarly, native gold is present in a few (one to four) particles in the flotation test feed. No gold was detected in the rougher tailings-leach residue which does not mean that there is none present. In general, the size of the gold particles is extremely small (four to 12 microns), only two or three particles reach 50 microns size. The native gold, according to its reflection color, contains about 70-80 percent Au and the rest is silver.

Cerargyrite (AgCl) the new name is Chlorargyrite) is a minor component. It occurs in the plus 100 mesh screen fraction and in the gravity concentrate of the ground hand sample of the "grit" (from trace to two volume percent respectively, see Table III). Similarly it is present in the same amount in the same screen and gravity products of the flotation test feed. The cerargyrite is a transparent mineral, it crystallizes in the cubic system hence it is isotropic which makes it difficult to detect in thin section. Its waxy sheen under reflected light, however, is more conspicous, thus it is easy to identify. The grain size of the cerargyrite ranges from 80 to 160 microns.

Steel Chips (tramp iron) are foreign components derived from the grinding media. They are more frequent in the ground hand sample of the "grit" and in the rougher tailings-leach residue than in the flotation test feeds screen and gravity products. They are relatively hard particles with moderately high reflectivity of white color. Thus they can be mistaken for gold by superficial observation (gold is soft, orange yellow with high reflectivity).

5. According to the above described microscopic examination the received samples show a simple mineralogy. Hard quartz and somewhat softer devitrified glass are the predominant constituents along with minor opaque minerals of which hematite and hydrous iron oxides are the most common ones. Sulfides and copper carbonate are present only in trace or very minor amounts. Native gold and silver chloride are the economically valuable minerals but their distribution is low.

6. The assays of the "high grade" ore and the flotation test feed showed a considerable difference between gold (0.617 oz/t) and silver (13.36 oz/t) values. This difference indicates that besides native gold (which may carry up to 20-25 weight percent silver as solid solution) some other discrete silver mineral(s) occur in the samples. Thus it was anticipated that some native silver, Ag sulfides or silver sulfosalts are the major silver carriers in the samples. The microscopic examination, however, did not show any of the above type silver minerals, instead silver chloride (cerargyrite) was found in medium size grains of low distribution. The cerargyrite is a high grade silver mineral carrying 65-70 weight percent Ag. It is soft, hence it comminutes rapidly in grinding and it is easily soluble in cyanide, but poorly in dilute hydrochloric acid. In most

I hope the report is satisfactory in clarifying the main question, what inhibits the recovery of the gold and silver.

Should you have any question concerning the above or any other mineralogical matter please do not hesitate to call me, I am more than glad to answer them.

Sincerely,

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László Dudás
Mineralogist

cases the cerargyrite is a supergene mineral, derives from some preexistent hypogene silver sulfosalts, hence it occurs in the oxidized zone of ore deposits. The cerargyrite is associated with other supergene lead, copper and particularly with iron oxide and hydrous iron oxide minerals. The circulating surface water carries many elements in solution. Besides lead, copper, zinc it can carry silver which may co-precipitate with the hydrous iron oxide impregnating the devitrified glass (also some of the discrete hydrous iron oxide grains, e.g. goethite, limonite, could contain certain amount of silver). Thus the hydrous iron oxides along with the cerargyrite (AgCl) are responsible for the silver content of the ore.

7. It was stated above (paragraph 3) that gold is fairly frequent in very small grain sizes and in low concentration. Small, four to 12 micron size free and locked particles were found in 15 out of the 24 polished sections from one to four particles per section. In only three cases has reached the particle size 50 microns. In the received samples the native gold occurs in three modes: (1) free particles, (2) locked with quartz, and (3) locked with discrete hydrous iron oxide (limonite).

8. The recovery of the gold either by flotation or cyanidation is greatly hindered by two factors: (1) overwhelming slimes produced by the devitrified glass, and (2) excessive impregnation of the devitrified glass by hydrous iron oxide solution. In the first case the fine free gold particles may get covered with fine slimes preventing its concentration in flotation. In the second case the deleterious hydrous iron oxides rapidly consume excessive amount of cyanide before it can reach the gold or silver particles or grains.

9. The necessity of fine grinding which would require a slime flotation circuit in the mill, and extreme cyanide consumption of the ore raises the question of economy in treating this ore. It appears that the best solution for utilizing this ore is to sell it as flux to a smelter.

Tables of volumetric percent distribution of the transparent and opaque mineral components are appended.

László Dudás

László Dudás
Mineralogist

TABLE I.

Volumetric Percent Distribution of the Transparent Mineral Components in the Hand Samples and Gravity Products of a Gold Ore from United Verde Extension Mine, Jerome, Yavapai Co., Arizona.

Names of Minerals — No.:	809	902		
	"Chert" As is	G As is	r i Gravity Conc.	t Tail
Weight %:			15.38	84.62
Quartz (fine grain "chert" free	60	27	26	29
Quartz (medium size, vein)	30	17	18	16
Quartz coated w. hydr.Fe oxides	2			
Quartz locked w. hydr. Fe oxides	8			
Quartz locked w. devitrified matrix				
Devitrified matrix (very fine)		9	8	10
" impregn. w. hydr. Fe oxides		33	30	35
Malachite				
Chlorite				
Opaques		14	18	10
Total	100	100	100	100

TABLE II.

Volumetric Percent Distribution of the Transparent Mineral Components in Number One Composites of Flotation Feed and Rougher Tailings - Leach Residue of a Gold Ore from United Verde Extension Mine, Jerome, Yavapai County, Arizona.

Test No.: 9.	Composite No. 1. Flotation Feed					Rough.Tail-Leach Residue		
	"As is"		Mixed Total	Gravity		"As is"	Gravity	
	+100	-100	-100	Conc.	Tail		Conc.	Tail
Weight %:	30.20	69.80		17.50	82.50		20.0	80.0
Names of Minerals — Assays: Au oz/t:			0.617			0.208		
Ag oz/t:			13.36			5.54		
Quartz free						24	23	30
Quartz (fine,"chert") free	20	25	18	10	17			
Quartz (med.size vein) free	24	15	14	18	15			
Quartz coated w. hydr. Fe oxides								
Quartz locked w. hydr. Fe oxides	4	2	3	6	3	tr	1	tr
Devitrified matrix (fine)	6	6	4	4	10	10	6	6
Devitrified matr.impregn.w.hydr.Fe ox.	14	40	41	30	41	48	42	44
Malachite	tr		tr	tr			tr	
Chlorite			tr					
Opaques	32	12	20	32	14	18	28	20
Total	100	100	100	100	100	100	100	100

TABLE III.

Volumetric Percent Distribution of the Opaque Mineral Components in the Hand Samples and Gravity Products of a Gold Ore from United Verde Extension Mine, Jerome, Yavapai County, Arizona.

Names of Minerals	809 "Chert"	902 Grit			
Mesh:		+100	-100		
	"As is"			Gravity Conc.	Tail
Weight %:		40.0	60.0	15.38	84.62
Transparent Gangue free	96	34	30	33	28
" locked w. hematite	2	1	tr	1	1
" " " hydr.Fe oxides	tr				tr
" " " pyrite	tr	tr	tr	1	tr
" " " rutile	1	1	1	tr	1
" " " gold				tr	tr
" impregn. w. hydr. Fe oxides	1	60	67	60	69
Hematite free			1		
Hydrous Iron Oxides free			tr	tr	
" locked w. gold					
Native Gold free				tr	tr
Pyrite					
Chalcopyrite					
Cerargyrite (AgCl) free		tr		2	
Steel Chips		4	1	3	1
Total	100	100	100	100	100

TABLE IV.

Volumetric Percent Distribution of the Opaque Mineral Components in the Number One Composites of Flotation Feed and Rougher Tailings - Leach Residue of a Gold ore from United Verde Extension Mine, Jerome, Yavapai County, Arizona.

		Composite No. 1.							
		Flotation Feed					Rough.Tail-Leach Residue		
		"As is"		Mixed Total	Gravity Conc.	Gravity Tail	"As is"	Gravity Conc.	Gravity Tail
	Mesh:	+100	-100	-100					
	Weight %:	30.20	69.80		17.50	82.50		20.0	80.0
Names of Minerals	Assays Au oz/t:			0.617			0.208		
	Ag oz/t:			13.36			5.54		
Transparent Gangue free		70	37	31	30	50	38	40	99
" locked w. hematite		6	3	2	8	2	3	2	tr
" " " Hydr. Fe oxides		2	2	1	6	1	2	2	tr
" " " pyrite		tr	tr	tr	tr	tr	1	1	
" " " rutile		2	1	1	3	1			
" " " gold		tr		tr	tr	tr			
" impreg." Fe oxides		14	54	60	41	45	45	41	tr
Hematite free		3	2	3	4	1	5	4	1
Hydrous Fe Oxides free		3	1	1	4	tr	3	2	tr
" locked w. gold					tr				
Native Gold free		tr	tr	tr	tr	tr			
Pyrite free		tr	tr	tr				2	tr
Chalcopyrite free					tr				
Cerargyrite		tr			2				
Steel Chips		tr	tr	1	2	tr	3	6	tr
Total		100	100	100	100	100	100	100	100