MOUNT NANSEN MINES LIMITED

COMPILATION OF MINERALOGICAL AND METALLURGICAL DATA

by

F. Bianconi - R. Saager

December 9, 1969

November 3, 1969

COMPILATION OF MINERALOGICAL REPORTS

(October 1965-April 1969)

by

R. Saager

COMPILATION OF MINERALOGICAL REPORTS

Discussing Ores from Huestis and Webber Mine

This summary is a compilation of the following 3 mineralogical reports carried out on the Webber and Huestis ore:

Mineralogical Investigation of Ore Samples from Mount Nansen Mines, Yukon Territory by W. Petruk, Canada Department of Mines and Technical Surveys, October 19, 1965.

Mineralogical Investigation of a Sample of Silver-Gold Ore from Mount Nansen Mines, Yukon Territory by D. Owens, Canada Department of Energy, Mines and Resources, August 18, 1968.

Mineralogical Investigation of Sulfide and Oxide Ore from Mount Nansen Mines Ltd. by R. Schmidt, Hazen Research Inc. Golden, Colorado, April 23, 1969.

Petruk (1965) investigated ore from the Webber Mine, Owens (1968) ore from Huestis and Schmidt (1969) ore from Huestis and Webber. All investigations have been carried out on a limited amount of material and altogether 40-50 polished sections were studied by the various investigators. Due to this rather poor statistic it is justified to assume that the obtained results are not representative for the Webber - Huestis ore.

The tree investigators reported the following ore-minerals: listed according to their approximate frequency.

Huestis:		Webber:	
Pyrite	FeS_2	Pyrite	FeS_2
Arsenopyrite	$FeAsS$	Arsenopyrite	$FeAsS$
Galena	PbS	Bindheimite	$Pb_2Sb_2O_7 \cdot nH_2O$
Sphalerite	ZnS	Anglesite	$PbSO_4$
Freibergite	$(Cu,Ag)_3 SbS_3$	Freibergite	$(Cu,Ag)_3SbS_3$
Jamesonite	$Pb_4FeSb_5S_{14}$	Galena	PbS
Bournonite	$CuPbSbS_3$	Freieslebenite	$Pb_3Ag_5Sb_5S_{12}$
Chalcopyrite	$CuFeS_2$	Argentite	Ag_2S
Bornite	Cu_5FeS_4	Silver	Ag
Pyrrhotite	FeS	Gold	Au
Argentite	Ag_2S	Covellite	CuS
Miargyrite	$AgSbS_2$	Sphalerite	ZnS
Boulangerite	$Pb_5Sb_4S_{11}$	Chalcopyrite	$CuFeS_2$
Gold	Au	Bornite	Cu_5FeS_4
Stephanite	Ag_5SbS_4	Bournonite	$CuPbSbS_3$

Huestis:		Webber:	
Covellite	CuS	Jamesonite	$Pb_4FeSb_5S_{14}$
Various Oxides in trace amounts		Tenorite	CuO
		Goethite	$Fe_2O_3 \cdot H_2O$
		Scorodite	$FeAsO_4 \cdot 2H_2O$
		Arsenolite	As_2O_3
		Massicot	Supergene Pb-Mineral
		Argentian-bindheimite ?	
		Argento-jarosite ?	
		Ag-antimonate ?	
		Stetefeldite ?	
		Partzite ?	
		Arsenobismite ?	

The above list for the two working indicates clearly the predominance of supergene and oxide minerals in Webber. Quite a few of the oxide minerals, however, must be regarded as guesses since they are very difficult to determine.

The following minerals already included in the above lists can be regarded as silver minerals i.e. they contain silver or can take silver into solid solution.

Freibergite $(Cu,Ag)_3SbS_3$	Cu can be replaced by Ag to a very large extent but not completely.
Argentite Ag_2S	This is a typical mineral of the oxide zones where it is formed by the break-down of other Ag-minerals such as galena or freibergite.
Galena PbS	Can contain up to 0.1% Ag_2S in solid solution. Maximum solubility of Ag_2S at the melting point of PbS is 0.6%. Higher Ag-contents must, however, be related to microscopic to submicroscopic inclusions of silver minerals such as freibergite, polybasite, pyrargyrite, argentite, etc. $AgBiS_2$, however, can go into solid solution with PbS up to an amount of 6% Ag, which might cause an anomalous bireflection.
Miargyrite $AgSbS_2$	Occurs very often together with other Ag-sulfosalts such as pyrargyrite, stephanite, etc.
Stephanite Ag_5SbS_4	Rather similar to miargyrite.

Freieslebenite $Pb_3Ag_5Sb_5S_{12}$	Similar to stephanite and miargyrite
Bindheimite $Pb_2Sb_2O_7 \cdot nH_2O$	Occurs as an alteration product and contains up to 1.1% Ag (Schmidt, 1969)
Ag-antimonate, partzite, stetefeldite, argentian-bindheimite, argento-jarosite	The presence of these secondary Ag-minerals has been assumed by Schmidt (1969) in the Webber ore since he found only residual freibergite and galena and only traces of native silver which cannot account for the relatively high silver content of this ore.

According to Ramdohr (1960) neither jamesonite nor bournonite, bornite and boulangerite can take Ag into solid solution and therefore cannot be regarded as silver minerals.

It, however, must be noted that they are usually intimately intergrown with galena and freibergite and therefore could contain inclusions of these two minerals.

Mineralogy

Pyrite:

It occurs usually as masses or coarse hypidiomorphic grains which are often mantled by sphalerite and/or galena. With arsenopyrite pyrite forms often combined grains. In the oxide-ore pyrite is replaced by goethite. The pyrite masses and grains vary in size from 50 microns to one centimeter. If pyrite occurs in boulangerite or bournonite its size varies from 20 microns to 0.8 microns (Petruk, 1968). Pyrite contains inclusions of various sulfides and also of gold; the size of these inclusions range from a few microns to 2 mm.

Arsenopyrite:

Owens (1968) reported less arsenopyrite then pyrite, its grain size being generally smaller. Individual grains and aggregates vary in size from a few microns to about 1.2 mm, while arsenopyrite inclusions in galena and freibergite are from 10 to about 350 microns in size. Petruk (1965) found in the Webber ore the arsenopyrite grains bordered and replaced by arsenobismite, scorodite, goethite and covellite. Schmidt (1969) regards arsenopyrite as one of the early minerals, which, however, has suffered much less from replacement than pyrite and sphalerite. Gold, freibergite, galena, sphalerite and pyrite have been found as inclusions in arsenopyrite ranging in size from 2 to 250 microns.

Chalcopyrite and sphalerite:

Owens (1968) regards sphalerite as one of the major constituents, which reflects the non-representativity of his samples. He observed sphalerite in massive form and in irregular aggregates having a size from 10 microns to 2 mm. In the Webber ore sphalerite is partially replaced by goethite and covellite. According to Schmidt (1969) sphalerite is heavily replaced by galena. Chalcopyrite is extremely seldom, but occurs in the usual form of exsolution plebs in sphalerite, very rarely it has been found as coarse to medium grained aggregates independent of the sphalerite mineralization.

Minor sulfosalts (boulangerite, bournonite, jamesonite):

These minerals have been reported by the three authors, however, it seems that some must have been incorrectly identified since none of the authors has reported all 3 ore minerals i.e. sometimes jamesonite must have been identified as bournonite or boulangerite or vice versa. Owens (1968) found less bournonite than galena, often it seems to be intergrown with freibergite and galena or occurs as combination with boulangerite or arsenopyrite. Grain size of bournonite is 0.05-6 mm. Inclusions in bournonite are rare. It is interesting to note that Schmidt (1969) did not find any bournonite in the Huestis ore. He reports it, however, from the oxide-ore where it directly alters to bindheimite. Owens (1968) found boulangerite in very small amounts as individual sheaf-like clusters or elongated grains. Some of the boulangerite is also intergrown with galena, sphalerite and bournonite. Inclusions seem to be rare. Schmidt (1969) did not report boulangerite but it seems that he identified this mineral as jamesonite, which in the sulfide-ore is relatively fine grained. It replaces most of the other sulfides, however, it seems that more than one stage of jamesonite mineralization has occurred. A jamesonite concentrate showed 0.58% Ag and 12.0% Pb. According to Ramdohr Ag goes not into the jamesonite molecule, the above content must therefore be related to inclusions of Ag-minerals. In the oxide-ore jamesonite alters as bournonite directly to yellow and greenish bindheimite. Anglesite sometimes shows an intergrowth with feathery textured bindheimite indicating a former galena-jamesonite assemblage. Petruk (1965) did not report any of the above sulfosalts.

Silver minerals (Galena, freibergite, argentite, freieslebenite, miargyrite, native silver):

Galena which actually is not a silver mineral, however, as pointed out earlier, is in fact an important silver carrier and has also been included in this group. Galena is the most abundant lead-bearing mineral, it occurs largely as masses and disseminations in gangue, pyrite and sphalerite. It is intergrown with arsenopyrite, freibergite, bournonite, pyrite and sphalerite. The grain-size of galena varies from 10 microns to 8 mm (Owens, 1968). Schmidt (1969) found that in the oxide-ore galena is directly replaced by supergene chalcocite and covellite, some of it may alter directly to anglesite forming the earlier mentioned intergrowth with bindheimite. Freibergite occurs as irregular masses, locally replacing sphalerite and intimately intergrown with galena and rarely with coarse chalcopyrite. Owens (1968) found relatively few inclusions in freibergite, he also reports intergrowth with galena, arsenopyrite, bournonite and sphalerite; the grain-size of freibergite being 10-750 microns. In the Webber oxide-ore freibergite occurs as a residual sulfide, and seems to alter to bindheimite or related substances. Miargyrite or pyrargyrite has been found in the sulfide ore in small amounts where it was found replacing galena. Freieslebenite was only reported by Petruk (1965) from Webber in grain-sizes from 10-200 microns. He might have confused it with freibergite. Argentite is reported by Petruk (1965) in grain-size from 1-100 microns. The larger grains contain inclusions of native silver. Native silver was also reported by Schmidt (1969) in the Webber ore as minute particles and spongy aggregates associated with earthy alteration products. He regards silver of secondary origin formed by the alteration of freibergite. The small amount of freibergite and the traces of native silver cannot account for the overall Ag-content of 1%. No halides were found in an NH_4OH extraction; bindheimite showed an Ag-content of 1.1% Ag. Schmidt, therefore, concluded that much of the silver present in the oxide-ore occurs as ill defined Ag-antimonates or oxides. He could not find any signs for the presence of argento-jarosite.

Gold:

Petruk (1965) did not find any gold-bearing minerals in his samples from Webber, probably due to bad polishing, Owens (1968) found 33 grains of gold. 9 occur as inclusions in arsenopyrite, 15 as inclusions in pyrite, 7 as inclusions in galena-gangue, 1 grain was found in bournonite and 1 grain in freibergite. Grain-size varies from 2-180 microns. Microprobe work revealed a fineness of 800. Schmidt found gold in abundance in areas characterized by fine grained pyrite, it furthermore is intergrown with galena, sphalerite, pyrite and arsenopyrite. Schmidt also found gold inclusions in pyrite and arsenopyrite. The grain-size of gold was determined at 5-500 microns, averaging 20 microns. In the oxide-ore Schmidt found native gold as minute particles in small quantity.

Conclusion

From the three reports the Petruk report covering the Webber ore is very poor and gives also very little informations, it furthermore seems that he confused many of the more difficult identifyable minerals. The two other reports are both good and complement each other, however, both are based on too little sample material and are therefore not representative. It seems that Owens and Schmidt identified the same mineral as boulangerite and jamesonite respectively. From the microphotographs it seems, however, that the mineral actually is jamesonite due to its crystallographic development. Owens report gives a lot of detail data from Huestis, which is not contained in the Schmidt report. His report from the Webber ore is rather interesting, however, does not contain many information on the various alteration products, mainly due to their extremely difficult identification.

Silver

In the sulfide-ore Ag is mainly contained in freibergite and galena, it furthermore occurs in miargyrite-pyrargyrite and native silver. These minerals are quite intensively intergrown with the other sulfides or with each other. The grain-size of galena and freibergite varies, but generally is comparatively large. Miargyrite-Pyrargyrite on the other hand is relatively very fine grained.

In the oxide-ore most of the galena and freibergite together with jamesonite (boulangerite ?) have altered to a rather loose intergrowth of anglesite and earthy bindheimite, the yellow-green mineral which occurs as coatings on the rock. The majority of the silver content apparently occurs in ill-defined Ag-antimonate or oxide or as argentian bindheimite, and also as spongy aggregates of native silver. No Ag-halides and no argento-jarosite was found. The oxidation has apparently occurred in situ with very little chemical transport except for Zn. Considering this lack of Ag-transport and the almost total absence of supergene Ag-minerals, it must be considered that the original Webber ore must have been richer in Ag-minerals than the present sulfide ore at Huestis.

Comparisons of the Pb/Sb ratios obtained by x-ray fluorescence from the Huestis sulfide ore (23) and the Weber oxide-ore (1.6) indicate also a higher jamesonite,boulangerite, bournonite and/or freibergite content in original unaltered Webber ore, provided no major chemical transport took place during the alteration.

Gold

In the Huestis ore Schmidt reports an abundance of native gold in areas characterized by fine grained pyrite. Gold is intergrown with galena, sphalerite, pyrite and arsenopyrite, and seems to be related to the galena mineralization. Gold also has been found as inclusions in pyrite and arsenopyrite, in this instance apparently belonging to an earlier gold mineralization. Grain-size for gold varies from 5-500 microns, the average being 20 microns. Owens reports a grain-size from 2 to 180 microns. He found 27% of the gold particles as inclusions in arsenopyrite, 45% as inclusions in pyrite, 21% as inclusions in galena and 7% as inclusions in bournonite or freibergite.

In the oxide-ore from Webber native gold was observed only as minute particles in small quantity.

File: Mt. Nansen Mines Limited

Subject: Metallurgical testing

November 5, 1969
Dr. F. Bianconi

SUMMARY REVIEW OF

METALLURGICAL TESTS ON MT. NANSEN ORE

(HUESTIS & WEBBER)

by

BRITTON RES. LAB.

1964

Oct. 22 Letter from L. G. White (Consulting Mining Engineer) to John Britton, to authorize him to start preliminary tests on Webber ore.

"It is understood that you will proceed immediately with straight cyanidation tests following which you intend to conduct preliminary flotation work to see whether the lead antimony content can be separated into a separate concentrate."

Oct. 30 First results on 2 cyanide tests on Webber ore. Gold recovery: 82.9, resp. 83.1%, silver recovery 87.4, resp. 89.1%. High cyanide and lime consumption.

Nov. 9 First results on flotation test on same ore. Flotation followed by cyanidation of tailings overall recovery, gold 80.0%, silver 93.6%. Compared with straight cyanidation the silver recovery fell slightly; cyanide consumption dropped considerably.

1965

May 28 Letter from B. S. Imrie to J. Britton confirming that Mt. Nansen desired metallurgical tests to be carried out on a bulk sample from Webber (ore from the 500 Raise, approx. 2,000 lbs and averaging 0.45 oz. Au/ton and 30 oz. Ag/ton).

"It is expected that the milling rate will be approx. 500 tons per day, although quite conceivably the milling rate could be started in the 200 ton range" (!)

July 12 Memo from B. S. Imrie to A. Stone. The results of two tests by Britton are anticipated (see August 2).

August 2 Summary on tests completed on bulk sample of Webber ore, received June 11.

Assay			
	Au	0.445	oz/ton
	Ag	32.6	oz/ton
	Pb	2.06	%
	Zn	0.15	%
	Cu	0.06	%
	Sb	1.29	%

Assay			
	As	2.52 %	
	Fe	5.96 %	(acid soluble)
	S	1.52 %	(total)
	S	1.27 %	(sulphate)
	S	0.25 %	(sulphide)

Ore highly oxidized: 16% of S as sulphide and the remainder as sulphate, including iron sulphate.

Cyanidation without previous treatment: excessive cyanide consumption (12 lbs/ton), iron salts in the ore react with cyanide.

Cyanidation after 15 hours pre-aeration in the presence of lime to oxidize the iron salts.

3 Tests

	63% - 200	87% - 200	94% - 200
Rec. Au	72.8%		74.7%
Rec. Ag	83.7%		85.4%
Time	72 h.		72 h.
			Consumption increased

"Present indications are that the best extraction that can be expected by direct cyanidation of the ore would be about 73% of the gold and 84% of the silver. These results could be obtained by aerating the ground ore in the presence of lime, thickening and possibly filtering the pulp, repulping, agitating with cyanide and lime solution for 72 hours, thickening and filtering. The cyanide and lime consumptions would be about 4 pounds of sodium cyanide and 20 pounds of calcium oxide per ton of ore. The recoveries could be increased by 3 to 4% by finer grinding and prolonged treatments but it is almost certain that the extra capital and operating costs would be more than offset the higher recoveries."

Sept. 25 Summary letter on latest results on same bulk sample from Webber.
Using sodium hydroxide or carbonate in place of lime during the pre-aerating and cyanidation stages, 82% Au and 87.1% Ag extracted in 72 hours. Cyanide consumption increased to 8.2 lbs/ton of ore.

At this point, Britton suspended work on the Webber ore and started on a sample from the Huestis vein.

Oct. 4 Memo from B. S. Imrie to file.
First results on direct cyanidation of Huestis ore. (See Feb. 23, 1966).

Oct. 21 Memo from B.S. Imrie to file.
Further tests on direct cyanidation of Huestis ore. (See Feb. 23, 1966).

1966

Feb. 23 Progress report No. 1, Tests on a sample of Gold-Silver ore from the Huestis orebody.

This report gives the results of a serie of tests performed on a Huestis sample (500 lbs. from Huestis 4300, 12 Vein).

In the summary it is stated:

"1. The sample used for the tests assayed 0.70 oz/ton of Au and 15.8 oz/ton of Ag, 1.38% lead, 1.82% zinc, 0.11% copper, 3.06% arsenic and 0.29% antimony. It was slightly oxidized.

2. Direct cyanidation, even after very fine grinding, recovered only 25% of the gold and 30% of the silver in the ore. Most of the remaining gold is believed to be intimately associated with arsenopyrite; the refractory silver is probably associated with galena.

3. Selective flotation (test H2) gave a silver-lead concentrate assaying 2.94 oz. Au and 272.5 oz. Ag per ton, 24.67% lead, 13.75% zinc, 4.73% antimony and 5.95% arsenic and a gold concentrate assaying 2.64 oz. Au and 2.9 oz. Ag per ton, 0.13% lead, 0.10% zinc, 0.12% antimony and 16.89% arsenic. Both of these concentrates should be saleable, but it would be desirable to upgrade them if possible in order to reduce freight and treatment charges.

4. When treating similar ore in a full-scale mill, it should be possible to recover at least 90% of the gold, 95% of the silver and 87% of the lead in the form of separate silver-lead and gold concentrates.

5. As an alternative to shipping the gold concentrate to a smelter, its treatment by roasting and cyaniding has been investigated. The results indicate that it should be possible to recover about 87% of the gold and 60% of the silver in the concentrate by a 2-stage treatment, involving roasting, cyaniding, high-temperature treatment of the residue and re-cyaniding. The gold and silver would be precipitated from the cyanide solutions and smelted at the mine or shipped to a smelter."

Assay of ore	Au	0.70	oz/ton
	Ag	15.8	oz/ton
	Pb	1.38%	
	Zn	1.82	
	Cu	0.11	
	As	3.06	
	Sb	0.29	
	Fe	8.07	
	S	7.52	(total)
	S	0.04	(sulphate)
	S	7.48	(sulphide, by difference)

Spectrographic (semi-quantitative)

Al	4.0%
Ba	0.02
B	0.01
Cd	0.02
Ca	2.0
Cr	0.001

-/4

Co	Trace
Mg	0.9
Mn	0.3
Mo	0.001
Ni	0.001
Si	High
Sn	0.003
Ti	0.1
V	0.0008

Be, Bi, Ga, Nb, Sr, Ta, W : not detected

Specific Gr.: 2.97, equiv. to 10.8 cu.f/ton.

Results of tests

Test B.C. 1, Direct Cyanidation

Grind:	80% - 200
Agitation:	18 hours with lime
Cyanidation for 72 hours	
Recovery:	20% Au
	27.5% Ag
Further cyanidation for 24 hours	
Add. Recovery:	1.1% Au
	1.0% Ag
NaCN consumption:	3.04 lbs/ton
CaO consumption:	14.4 lbs/ton

Test B.C. 2, Direct Cyanidation

Grinding:	99% - 200
Cyanidation for 72 hours	
Recovery:	22.7% Au
	27.7% Ag
Further cyanidation for 72 hours	
Add. Recovery:	2.0% Au
	2.3% Ag
NaCN consumption:	4.64 lbs/ton
CaO consumption:	16.4 lbs/ton

Test H 1, Flotation

Grinding: 70% - 200

Flotation of two concentrates:

1) Silver - lead concentrate: 32% Au
90% Ag recovered

2) Pyrite - arsenopyrite concentrate:
60% Au
7% Ag recovered

Total recovery:	92% Au
	97% Ag
Concentration ratio:	3.5/1

The concentrates were not cleaned.

-/5

Test H2, Flotation, cyanidation and roasting

Test H2 involved the following stages of flotation:

1- Grinding (20% -200 mesh)
2- Ag-Pb conditioning
3- Ag-Pb rougher flotation
4- Ag-Pb cleaning
5- Ag-Pb recleaning
6- Zn conditioning
7- Zn rougher flotation
8- Zn cleaning
9- Au concentrate conditioning
10- Au concentrate rougher flotation
11- Au concentrate cleaning
12- Au concentrate recleaning

Results

a) Silver - lead rougher concentrate:

Recovery:	21.5% Au
	83.6% Ag
	83.2% Pb
Concentration ratio:	19.7/1

Most of the zinc floated with the silver and lead.

b) "Zinc concentrate":
only 0.6% of the zinc recovered.

c) pyrite/arsenopyrite concentrate:

Recovery:	46.1% Au
	2.2% Ag
Concentration ratio:	8.25/1

Overall recovery:	67.6% Au
	85.8% Ag
	83.2% Pb
Overall concentration ratio:	3.8/1

Cyanidation of gold concentrate

Test B.C. 3 (unroasted concentrate)

The pyrite/arsenopyrite (gold) concentrate was cyanided for two periods of 24 hours each. The residue was ground to 99% -325 mesh and cyanided for 24 hours.

Recovery:	3.6% Au only
	50.7% Ag
Tot. consumption:	NaCN 6.8 lbs/ton of concentrate
	CaO 27.6 lbs/ton of concentrate

Test B.C. 4 (cyanidation after roasting)

The pyrite/arsenopyrite concentrate was roasted under conditions which simulated as closely as possible the conditions used in a fluidised bed roaster. Calcine cyanided for 2 periods of 24 hours each; the residue was ground to 76% -325 mesh (85% -200) and cyanided for 24 hours. Residue was then treated at approximately 950°C for 30 minutes, ground to -28 mesh and re-cyanided.

Recovery:		76.1% Au
		54.3% Ag
Losses (roasting):		16.8% Au
Consumption	NaCN	5.2 lbs/ton of concentrate
	CaO	26.8 lbs/ton of concentrate

Referring to the losses, Britton states that "in practice, mechanical and roasting losses should not exceed 5% if the roaster gases are passed through an electrostatic precipitator whilst still hot and the collected dust is treated by cyanidation in the presence of activated carbon."

Assuming only 5% of loss then the recoveries would be as follows:

86.9% Au
59.8% Ag

Britton also anticipates the results of cyaniding the calcine, followed by high-temperature treatment of the residue and re-cyaniding. "Of the total gold in the calcine, 74% should be recovered by direct cyanidation and a further 17% by cyanidation after re-heating, giving a total recovery of 91%. 63% of the silver should also be recovered."

Test H 3, Flotation of a silver-lead concentrate after finer grinding.

In a attempt to upgrade the silver-lead concentrate, the ore was ground to 81% -200 mesh. The cyanide and zinc sulphate additions during grinding and cleaning were doubled in order to assist depression of the sphalerite. The rougher concentrate was cleaned three times.

Results

Grinding:	81% -200
Recovery:	12.6% Au
	77% Ag
	80.5% Pb
Concentrate ratio:	27.4/1

Comments

"The final silver-lead concentrate assayed 2.28 oz/ton Au, 345.9 oz/ton Ag, 32.93% Pb and 9.37% Zn. The silver and lead assays were about 1/3 higher than in test H2 and the zinc assay was 1/3 lower. Most of the improvement was due to the additional cleaning step rather than finer grinding."

The recovery dropped from 21.5% to 12.6% for Au and from 83.6% to 77% for Ag. These losses are probably due to the additional cleaning.

March 14 Britton sent two hand-written tables with the results of a further flotation test,

Test H 4. Same samples (Huestis ore) as in Feb. 23, 1966.

Stages
- Grinding (70% -200 mesh)
- Conditioning
- Ag-Pb rougher flotation
- Regrinding of +325 mesh rougher concentrate
- Ag-Pb cleaning
- Ag-Pb recleaning

Results

Recovery:	7.6% Au
	70.8% Ag
Concentrate ratio:	38.2/1

Oct. 28 Mt. Nansen sent 9 ore samples to Britton (5 Webber, 4 Huestis). With these samples a new serie of tests was started. The results of the tests were sent to Mt. Nansen as progress reports during 1967 and as a final comprehensive report (Dec. 29, 1967).

Nov. 23 Letter from J. Britton to A. T. Griffis with a summary of the proposed metallurgical methods and the expected recoveries in a full-scale operation.

Method

1) Oxide-ore

- crushing
- grinding
- aeration followed by thickening
- cyanidation
- 2-stage filtering
- precipitation
- smelting of precipitate to make gold-silver bullion

2) Sulphide ore

- crushing
- grinding
- flotation of silver-lead-zinc concentrate foolowed by cleaning of the concentrate
- flotation of gold-arsenopyrite-pyrite concentrate followed by cleaning
- 2-stage roasting of the gold concentrate in a fluidised bed roaster
- addition of the calcine to the agitators used for cyaniding the Webber ore, in order to extract the gold and silver.

Results which can be expected:

1) Oxide-ore:

Gold recovery:	(in bullion) 73%
Silver recovery:	(in bullion) 84%

2) Sulphide ore

Gold recovery:	78.6% (63.8% in bullion 14.8% in Ag-Pb-Zn-concentrate).
Silver recovery:	90.8% (4.8% in bullion 86.0% in Ag-Pb-Zn-concentrate).

In addition, 86% Pb and 88% Zn would be recovered in the concentrate.

Dec. 9 Test H 5, Flotation of "gold-arsenopyrite"-concentrate

Same sample from the Huestis ore as in the tests reported Feb. 23, 1966.

Test H 5 investigated the possibility of obtaining a high grade arsenical gold concentrate.

- Ore ground to 70% -200 mesh
- Ag-Pb concentrate: 45.7% Au recovery (concentrate not cleaned, therefore concentration ratio of 6/1 !)
- Addition of calcium hydroxide and ammonium chloride for the purpose of activating the pyrite and maintaining the arsenopyrite in a depressed condition; 4 concentrates removed in succession.
- Addition of sulphuric acid and collector; 5 concentrates removed.

49.8% Au was recovered in the combined arsenopyrite-pyrite concentrates, giving a total recovery of 95.5% Au.

From the different concentrates it appears that gold is associated with arsenopyrite and little, if any, is associated with pyrite. In this specifical sample there is a constant ratio gold: arsenopyrite as indicated in the different concentrates of about 0.075 to 1. "A satisfactory separation of pyrite and arsenopyrite was not achieved."

1967

Aug. 23 Memo from B.S. Imrie to Dr. D. D. Campbell. "It is becoming apparent that probably the best way to handle the ore would be by flotation to make a bulk concentrate. This would give us a simple flow sheet, and we would probably not have to bother about a cyanide plant at the present time. It appears that our concentrate should be saleable. It might be preferable to withhold the highly oxidized Webber ore portions until a later date when they could be treated with Brown-McDade ores, if they are compatible."

At this time, Mt. Nansen was handling the purchase of a second-hand mill from Minoca Mines, Vancouver Island (the agreement was signed Oct. 25, 1967); at the same time Britton reduces the metallurgical tests along the guidelines he used until then (selective flotation of 2 concentrates, Ag-Pb concentrate and a "gold concentrate") and starts a new serie based on the feasibility of a bulk concentrate (report of Dec. 29, 1967). It seems that this new concept was dictated by the purchase of the second-hand mill: the metallurgical tests had to change in order to find a process which would fit into the given mill !

Aug. 30 Dr. D. D. Campbell in a letter to Dr. Jerome replies to the memo of Mr. Imrie and he points out that the concept of the flotation of a bulk concentrate is wrong, as well as the proposed mill site; he also warns that such a flow sheet, incapable to treat the oxide-ore, would seriously affect an efficient operation.

Dec. 29 Tests on samples of Gold-Silver Ore from the Huestis and Webber orebodies submitted by Mount Nansen Mines Limited.

Final report on the metallurgical tests performed on the 9 samples of Webber and Huestis ore (see Oct. 28, 1966).

From the 9 samples, 3 composite samples were mixed for the metallurgical tests:

		Huestis 67A	Webber 67B	Huestis and Webber 50% of 67A + 50% of 67B
Gold	oz/ton	0.50	0.42	0.46
Silver	oz/ton	29.1	34.5	31.8
Lead (total)	%	0.50	2.45	1.48
Lead (oxide)	%	0.04	0.51	0.28
Zinc (total)	%	0.81	0.38	0.60
Copper	%	0.09	0.15	0.12
Arsenic (total)	%	4.25	2.27	3.26
Arsenic (oxide)	%	0.13	0.56	0.35
Antimony	%	0.68	0.90	0.79
Iron	%	5.98	5.22	5.60
Sulphur (total)	%	4.63	3.45	4.04
Sulphur (sulphate)	%	0.02	0.65	0.34
Degree of lead oxidation*	%	8	21	19
Specific gravity		2.85	2.64	2.74
Tonnage factor		11.2	12.1	11.7

$*\frac{\text{Oxide Pb \%}}{\text{Total Pb \%}} \times 100$

Gold and silver distribution in ground ore

1) Huestis composite 67A

Fraction	Weight %	Distribution % Au	Ag
+100 mesh	10.2	8.0	4.6
-100 +150 "	14.5	8.0	9.8
-150 +200 "	12.1	9.3	11.1
-200 +325 "	15.0	19.2	16.6
-325 "	48.2	55.5	57.9

2) Webber composite 67B

Fraction	Weight %	Distribution % Au	Ag
+100 mesh	5.6	2.7	2.1
-100 +150 "	12.2	6.2	7.5
-150 +200 "	11.7	8.1	9.3
-200 +325 "	15.5	14.4	15.5
-325 "	55.0	68.6	65.6

From the two tables it appears that most of the gold and of the silver are present in the finer fractions: in the Huestis sample 74.7% Au and 74.5% Ag are present in the -200 mesh fraction; in the Webber sample 83.0% Au and 81.1% Ag are present in the -200 mesh fraction. In the Huestis sample 8.0% Au is coarser than the +100 mesh.

Results of tests

Test 152-C1, Cyanidation of Composite 67A (Huestis)

The ground ore was jigged, the jig tailing was thickened, pre-aerated with lime for 16 hours. Cyanidation in 3 stages, each of 24 hours.

Results

Jig concentrate	8.2% Au recovery
	0.9% Ag "
Concentration ratio	50/1
Cyanidation	8.4% Au recovery
(72 hours total)	22.2% Ag "
Overall recovery	16.6% Au
	23.1% Ag
Consumption	NaCN 2.9 lbs/ton ore
	CaO 17.0 lbs/ton ore

Test 152-C2, Cyanidation of Composite 67B (Webber ore)

Same procedure as 152-C1.

Results

Jig concentrate	2.1% Au recovery
	0.9% Ag "
Concentration ratio	35/1
Cyanidation	46.9% Au recovery
(72 hours total)	48.3% Ag "
Overall recovery	49.0% Au
	49.2% Ag
Consumption	NaCN 7.8 lbs/ton ore
	CaO 18.1 lbs/ton ore

Comments

"It is evident from the above results that cyanidation would not be a suitable method for recovering the gold and silver from either the Huestis or the Webber ore."

However, it should be noted, that the Webber samples used in this serie of tests are not representative, since their "oxidation grade" is only 19%.

The tests from August 2, 1965, performed on a more oxidized ore showed much better recoveries and less reagents consumptions.

Test 152-1, Selective Flotation, Huestis 67A

The ground ore was jigged; the jig tailing was thickened and conditioned with lime and calcine. A silver-lead-zinc concentrate was then floated in 3 stages; the tailing was conditioned with sulphuric acid and more copper sulphate and a gold-arsenopyrite-pyrite concentrate was taken off in 3 stages.

Results

Grinding:	63% -200
Jig concentrate:	8.4% Au recovery
	0.8% Ag "
Concentration ratio:	38.5/1

Silver-lead flotation concentrate

	11.4% Au recovery
	86.2% Ag "
	78.0% Pb "
	71.5% Zn "
Concentration ratio	20/1

Gold concentrate

	74.5% Au recovery
	9.1% Ag "
Concentration ratio:	5.4/1

Overall recovery:	Gold	94.3%
	Silver	96.1%
	Lead	90.3%
	Zinc	94.2%
	Arsenic	89.5%
Concentration ratio:		4.2/1

Comments

Although the overall concentration ratio amount to only 4.2/1, the overall recoveries are very good. The silver-lead concentrate has a high grade (507.8 oz. Ag/ton) and a good concentration ratio (20/1). The gold concentrate has a concentration ratio of only 5.4/1 and needs a further concentration in order to increase the overall concentration ratio.

Test 152-2, Selective Flotation, Webber 67B

Conditioning with lime and cyanide, three stages flotation of silver-lead. Tailing conditioned with copper sulphate and sulphuric acid, 2 stages flotation of gold concentrate.

Results

Grinding:	71% -200
Silver-lead concentrate:	34.7% Au recovery
	69.0% Ag "
Concentration ratio:	10.2/1
Gold concentrate:	32.6% Au recovery
	10.7% Ag "
Concentration ratio:	12.7/1
Overall recovery:	67.3% Au
	79.7% Ag
Overall concentration ratio:	5.65/1

Comments

The selective flotation is not the suitable process for the Webber ore, since the recoveries are moderate and the concentration ratio very poor.

Test 152-4, Selective Flotation, Webber 67B

Flotation of silver-lead concentrate in 3 stages, of a gold concentrate and of 2 additional concentrates after addition of sodium sulphide in order to activate the oxidized lead and arsenic minerals. A final concentrate was taken off after adding oleic acid, in order to float oxide minerals.

Results

Grinding:	66% -200
Silver-lead concentrate:	69.9% Au recovery
	80.3% Ag "
Concentration ratio:	7.4/1
Gold concentrate:	5.6% Au recovery
	2.9% Ag "
Concentration ratio:	24.8/1
"Oxide" concentrate:	10.4% Au recovery
	8.1% Ag "
Concentration ratio:	9.3/1
Overall recovery:	85.9% Au
	91.3% Ag
Overall concentration ratio:	3.5/1

Comments

The addition of a third flotation stage, in order to float the oxides, involves an increase of the recoveries of both gold and silver, but the overall concentration ratio decreases to the very bad amount of 3.5/1.

Test 152-3, Bulk Flotation, Huestis + Webber, 67A + 67B.

Two rougher concentrates were taken off, followed by a scavenger concentrate. The rougher concentrates were combined and cleaned once.

Results

Grinding:	64% -200
1+2 rougher concentrates:	77.3% Au recovery
	87.2% Ag "
Concentration ratio:	5.3/1
Cleaned concentrates:	63.4% Au recovery
	77.5% Ag "
Concentration ratio:	9.15/1

Comments

The recoveries are moderate in both the cleaned and uncleaned concentrates. Cleaning of the concentrates involves a better concentration ratio but a loss in the recoveries of 14% for Au and 10% for Ag. However, most of this loss should be recovered by recirculating the cleaner tailings.

Test 152-5, Sulphide-Oxide Flotation, Huestis & Webber 67A + 67B

Sulphides floated in six stages; the combined concentrates were cleaned once. The sulphide rougher tailing was conditioned with oleic acid and an oxide concentrate removed and cleaned once.

Results

Grinding:	64% -200
Sulphide rougher concentrate	
Recovery	89.7% Au
	91.8% Ag
Concentration ratio:	3.9/1
Same cleaned	
Recovery	82.8% Au
	86.4% Ag
Concentration ratio:	6.1/1
Oxide concentrate	
Recovery	3.8% Au
	3.3% Ag
Concentration ratio:	13.7/1
Same cleaned	
Recovery	1.6% Au
	1.6% Ag
Concentration ratio:	60/1
Overall recovery:	
Concentrates not cleaned	
Recovery	93.5% Au
	95.1% Ag
Concentration ratio:	3/1
Concentrates cleaned	
Recovery	84.4% Au
	88.0% Ag
Concentration ratio:	5.6/1

Comments

Here again, the uncleaned concentrate shows good recoveries and a very poor concentration ratio (3 !), while the cleaned concentrate has a better concentration ratio (almost twice as much) and a loss of 10% Au and 7% Ag; these losses should be recovered almost completely by recirculating the cleaner tailings.

Cyanidation of test 152-5 oxide concentrate

Sample of cleaned oxide concentrate cyanided for three periods of 24 hours each, the pulp being filtered and the residue washed at the end of each stage.

Results

Time	72 hours
Recovery	53.0% Au
Consumption	NaCN 16.3 lbs/ton of conc.
	CaO 27.5 " "

Comments

"The high cyanide consumption was due mainly to the presence of soluble copper minerals in the concentrate. In view of the relatively low recoveries of gold and silver in both flotation and cyanidation, together with the high cyanide consumption and consequent fouling of the solution, the production and cyanidation of an oxide concentrate does not appear attractive."

Test 152-6, Sulphide-Oxide Flotation, Huestis and Webber, 67A + 67B.

A bulk rougher concentrate was floated in four stages and was cleaned twice; sodium sulphide and oleic acid were used during rougher flotation in order to improve the recovery of oxide minerals.

Results

Grinding:	64% -200
Rougher concentrate:	85% Au
	91% Ag
Concentration ratio:	4/1
Cleaned concentrate:	77.9% Au recovery
	82.3% Ag "
Concentration ratio:	6.6/1

Comments

"Most of the gold and silver dropped in cleaning should, however, be recovered by recirculating the cleaner tailings".

At the end of his report, Britton anticipates the results for full scale milling, based on the results of tests 152-5 and 152-6. The mill would produce a bulk concentrate cleaned twice.

Concentration ratio: 6/1

		Assays		Recoveries
		ore	concentr.	%
Gold	oz/ton	0.40	2.33	84
Silver	oz/ton	32.0	170	88
Lead	%	1.5	5.7	63
Zinc	%	0.6	3.1	87
Arsenic	%	3.3	16	80
Antimony	%	0.8	2.4	50
Copper	%	0.12	0.6	85
Iron	%	5.6	20	60
Sulphur	%	4.0	20	84
Bismuth	%	-	0.005	-
Acid insol.	%	-	26	-
Chlorine	%	-	0.06	-

1968

Jan. 26 Letter from D.D. Campbell to Mr. Britton. He asks for more specific tests, the following:

- Flotation at finer grinds than used to date;
- More tests on the Huestis type ore;
- Bench roast tests on the concentrates.

March 18 Report from Britton:

Tests on samples of Gold-Silver Ore from the Huestis and Webber Orebodies submitted by Mount Nansen Mines Limited.

1) Test 152-7, Flotation of Huestis, 67A

Grinding time increased from 30 to 40 minutes: 78% of the product was finer than 200 mesh. Ground ore treated by bulk flotation and concentrate cleaned twice.

Results

Bulk concentrate, cleaned twice:

Recovery	90.6% Au
	95.5% Ag
Concentration ratio:	5.4/1

Bulk concentrate, not cleaned:

Recovery	92.0% Au
	96.6% Ag
Concentration ratio:	3.5/1

Comments

"Allowing for the recirculation of the cleaner and recleaner tailings, the following results can be expected when treating similar ore in a full-scale mill:

Concentration ratio:	5.35/1
Recovery	93% Au
	97% Ag

Final concentrate high in siliceous matter (24.93% acid-insoluble matter), especially in the coarser fraction (66% acid-insoluble matter). Britton recommends further tests on regrinding of the rougher concentrate to about 90% -325: "if the acid-insoluble content of the concentrate could be reduced to say 12%, as expected, the weight of concentrate to be shipped and smelted would be reduced by about 15%.

2) Roasting tests

Samples of the final concentrates from tests 152-6 and 152-7 were roasted at a maximum temperature of 650°C.

Results

Roasting of 152-6 concentrate reduced the weight by 22.3%. The As assay was reduced from 16.04% to 2.57%, the Sb assay from 2.38% to 1.15% and the S assay from 21.42% to 2.75%. Losses were 3.5% for Au and 0.2% for Ag.

Roasting of 152-7 concentrate reduced the weight by 24.9%. The As assay was reduced from 17.39% to 1.87%, the Sb assay from 1.67% to 0.76% and S from 20.93% to 1.70%. Losses were 2.5% for Au and 1.6% for Ag.

-/16

3) Leaching tests (preliminary)

A sample of 152-7 concentrate was leached with a solution containing 60 - 62% Na_2S, at a temperature of 95 - 100°C and with the pulp stirred for a period of 1 hour.

Results

72% of the Sb in the concentrate was dissolved, leaving a residue assaying 0.47% Sb (1.67% before leaching). Losses: 1.7% in the weight, 0.1% Au and 0.3% Ag.

Comments

"As an alternative to leaching the bulk concentrate, the possibility of selective flotation of a silver-lead concentrate, followed by leaching of the silver-lead concentrate only should be considered. This concentrate would have an appreciably lower weight than the bulk concentrate, possibly less than 5%. The gold-arsenic concentrate would probably assay less than 0.5% Sb. It is expected that the antimony could be recovered from the leach solution in a saleable form. If this proves to be the case, the recovery of antimony would at least partly offset the cost of the leaching operation."

1969

May 21 Letter from Britton to E. Livgard (S and N, Mine Management Consultants Ltd.); he proposes further investigations of the Mount Nansen ore and also of the Brown-McDade property along these lines:

1) Flotation of a gold-silver-lead concentrate for shippment to a lead smelter;

2) Flotation of a gold-arsenopyrite concentrate, the pyrite being depressed;

3) Flotation of the pyrite and oxide minerals, followed by cyanidation;

4) Alternative to 3) - Cyanidation of the whole of the tailings from 2).
"Most of the laboratory work would be done on a mixture of equal weights of the three types of ore (Huestis, Webber, Brown-McDade) but some work would be done on the separate samples to ensure that the treatment of each is economic and to check whether possible variations in the proportions of the various ores would have a harmful effect on the results".

File: Mt. Nansen Mines Limited

November 5, 1969
Dr. R. Saager

Subject: Metallurgical testing

SUMMARY REVIEW OF

METALLURGICAL TESTS ON MT. NANSEN ORE

(HUESTIS & WEBBER)

by

DEPARTMENT OF ENERGY, MINES AND RESOURCES, OTTAWA

1965

April 2	Imrie establishes contact with L. E. Djingheuzian of the Mineral Processing Division, he furthermore inquires about the sample size needed for a complete metallurgical test.
April 23	Imrie organizes for a 600 pound sample of Webber ore to be shipped to Ottawa.
May 20	A 760 pound sample has been received and acknowledged by R.W. Bruce of the Non-Ferrous Minerals Section at Ottawa.
August 12	R. W. Bruce advices Imrie that tests on the Webber ore should commence within a week.
Nov. 4	Preliminary results have been submitted by Ottawa. They carried out Amalgamation, Cyanidation and Flotation tests. All these tests have been summarized in a later comprehensive report.
Dec. 30	The mineralogical report: "Mineralogical Investigation of Ore Samples from Mount Nansen Mines, Yukon Territory", has been received.

1966

Dec. 6 — A brief summary of all the tests carried out by Ottawa gives the following results:

60 - 70% -200 mesh grind
24 hours cyanidation
Approx. 82% recovery for silver and gold.

1967

July 4 — The comprehensive report from Ottawa entitled "Recovery of Gold and Silver from Mount Nansen Mines Limited, Carmacks Area, Yukon Territory" by T. F. Berry was received.

Test 1

Amalgamation showed extremely poor recoveries: 1% Au and 4.1% Ag, indicating virtually no free milling gold or silver in the ore.

Test 2-25 Straight cyanidation.

The 25 tests indigated that best results are obtained with a 24 hours agitation time and a 77.6% -200 mesh grind. Longer agitation times do not increase the recoveries substantially. For the above agitation time the recoveries are about 83% for Au and 86% for Ag. Consumptions lie at 6.76 lbs NaCN/t and 13.6 lbs CaO/t. Screen and infrasizer tests indicated that with a -10 microns grind 81% Au and 91.8% Ag might be recovered.

Test 27 Gravity and Flotation tests.

-10 mesh jigging resulted in a 2% Au and 9.8% Ag recovery, tabling of the jigtailings resulted in a 3.2% Au and 13.5% Ag recovery. The overall recovery including flotation was 25.1% for Au and 57.3% for Ag, straight flotation gave similar results. This approach was therefore considered unsuccessful.

Test 29 Cyanidation and Flotation

These tests were only partially successful as only an additional 4.7% of the gold and 5.8% of the silver was recovered, the total recovery being 77.2% Au, 86.1% Ag.

These results are in fact worse than straight cyanidation since the concentration ratio for the latter is > 100 as against 9.5 for the combination cyanidation, flotation.

Disliming of ground ore to remove barren material was unsuccessful.

1968

Feb. 15 Letter to Ottawa inquiring if arrangements could be made to have additional testing done on Mount Nansen sulphide ores. Imrie considers a sample grading 0.50 oz. Au and 25 oz. Ag as representative for Huestis !!

Feb. 16 Letter to J. Convey, Ottawa. Imrie suggests grinding tests at 65% -200 mesh and at 80% -200 mesh; after that bulk flotation tests with one-stage cleaning.

Feb. 20 S. Gray estimates, based on Britten Test 152-6, a net profit for a 50:50 Webber/Huestis ore of $25.00 per ton of ore treated, hear hear.

Feb. 23 Three boxes of Huestis and Webber ore have been air fraighted to Ottawa.

Feb. 29 Letter by J. G. Brady, Ottawa, in which he points out that the Webber ores do not respond to flotation concentration, whereas the sulphide ore (Huestis) might be ammenable to flotation.

March 4 Imrie writes back to Ottawa: "It should be appreciated that these samples will make a mixture of oxide and sulphide ores in that proportion which we expect to start milling late in August 1968."

March 6 1st Progress Report

A screen test on a combined sample 40% Webber and 60% Huestis indicates that a grind between 85% and 90% -200 mesh will be necessary to ensure best recovery for gold and silver.

Test 10

With a grind of 90% -200 mesh and a flotation time of 28 minutes the removal of seven separate concentrates gave a recovery of 83.1% for Au and 93.8% for Ag, but an extremely poor concentration ratio of 2.1. The size analysis of the flotation tailing shows 32.1% of the gold and 22.3% of the silver in the -10 micron fraction. Ottawa thinks, therefore, that it is doubtful whether one can recover an appreciable amount of this gold and silver.

June 18 2nd Progress Report

Flotation tests 11-17: Two grinds 75% -200 and 90% -200 were investigated and no difference in the overall recovery experienced. In test 20 the rougher concentrate was reground but the recovery could not be improved (test 20) at a 75% -200 initial grind. Test 22: Two stage flotation and cyanidation was employed, no scavanger was used. Obtained recovery was quite good and concentration ratio reasonable.

Based on a 75% -200 mesh grind:

	93.1% Au and 96.3% Ag recovery
Concentration ratio:	10.5

Based on a 90% -200 mesh grind:

	93.5% Au and 96.5% Ag recovery
Concentration ratio:	11.6

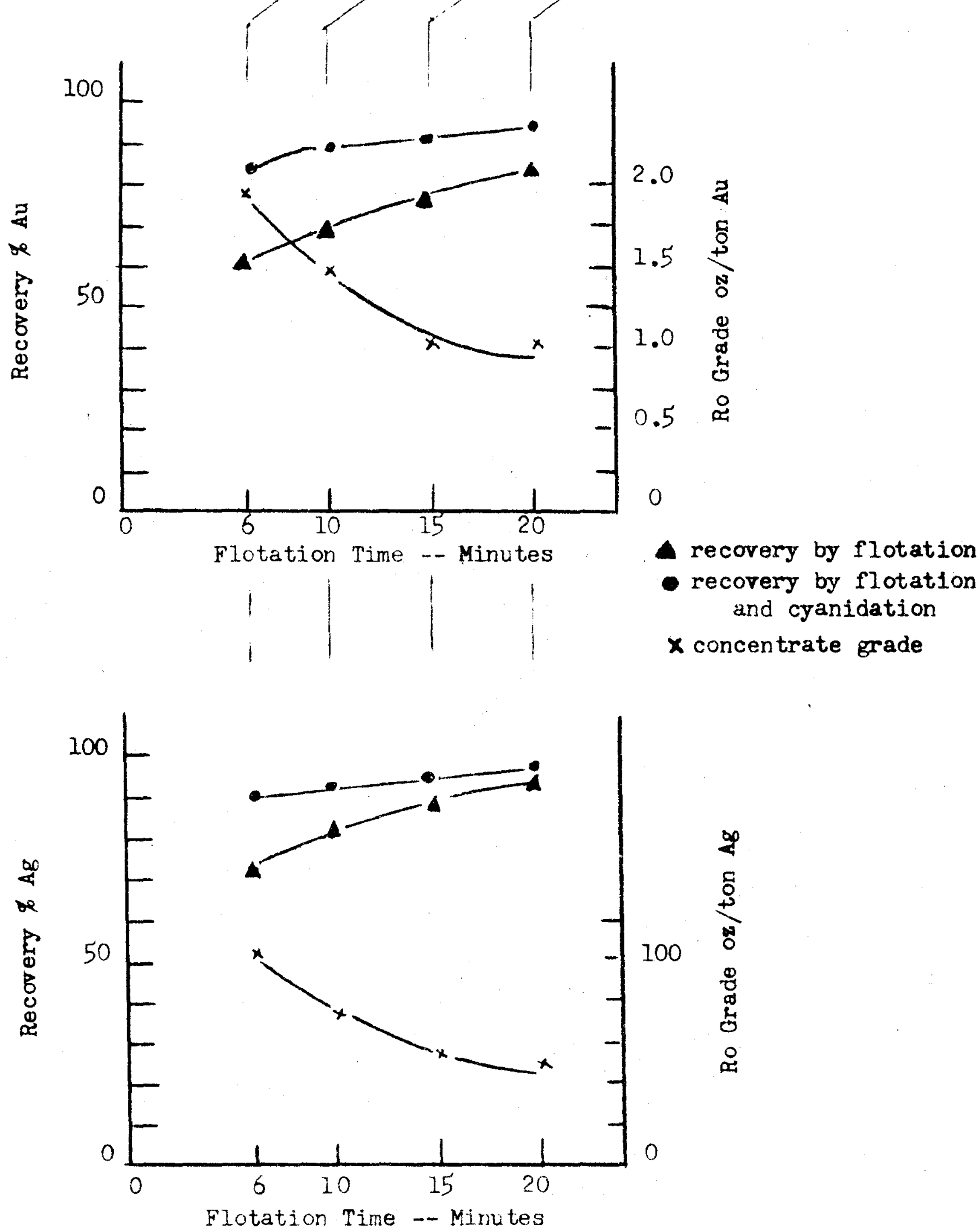
July 25 3rd Progress Report

Tests were carried out on the recoveries gained by grinding the ore to different finenesses. The results quite clearly showed that very little is gained by grinding finer than 55% -200 mesh. A flotation time of about 20 minutes ensures the optimum flotation recovery of Ag and Au. Together with cyanidation of the flotation tailing an overall recovery of about 94% of the Au and 96% of the Ag should be obtained. Concentrate not cleaned 28.1% = 3.6, concentrate cleaned 8.2% = 12.2 (Test 29).

Figure 1 indicates concentration grades against flotation time for test 26 - 29.

FIGURE 1 - Recovery and Grades Vs Flotation Time Tests 26-29

Reagent Consumption lb/ton Feed	NaCN -	1.20	0.96	0.72	0.56
	CaO -	6.72	6.28	6.16	6.00

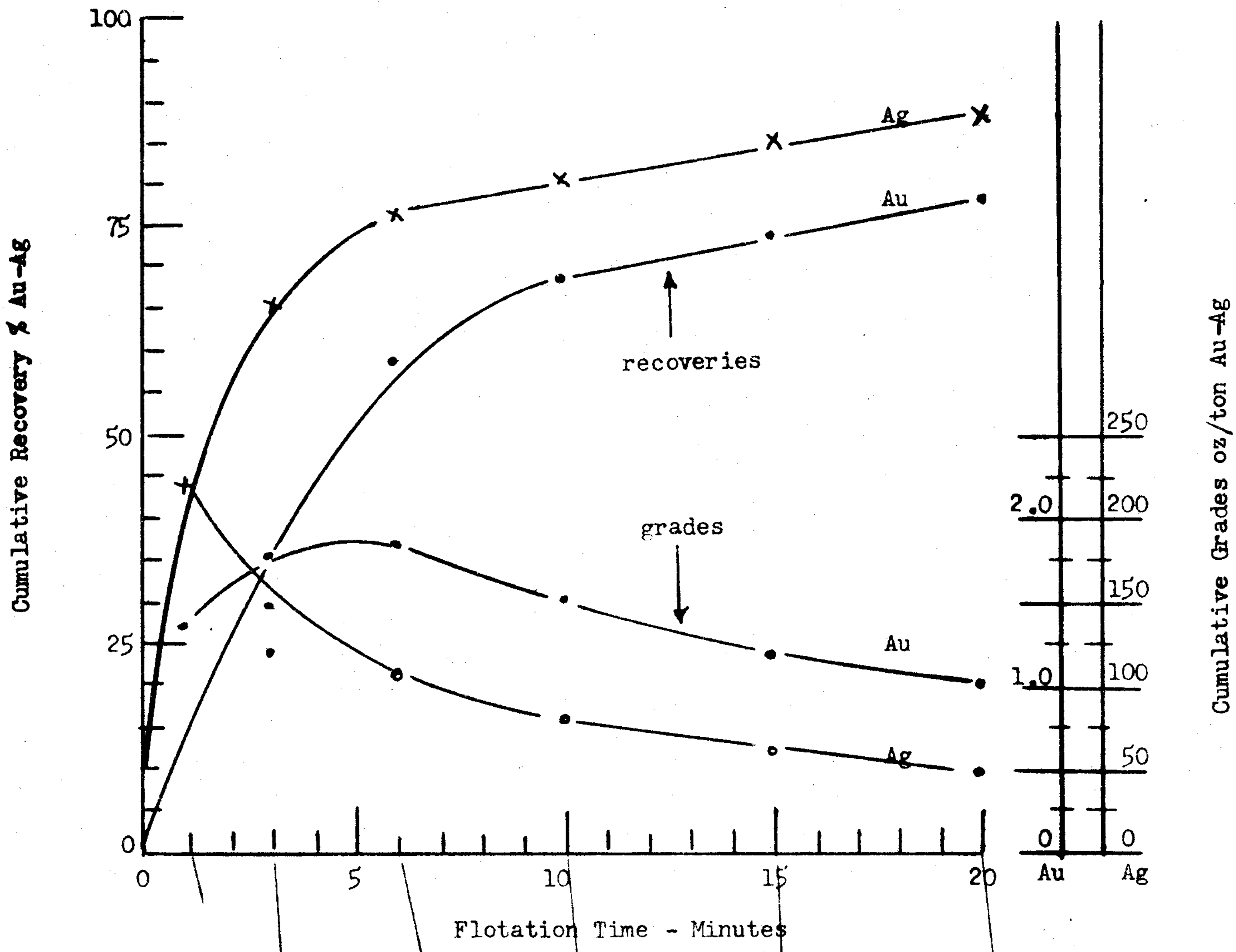


Flotation Reagents lb/Feed	301--	0.10	0.20	0.20	0.25
	Z-6--	0.15	0.20	0.30	0.40
	P.O.--	0.16	0.18	0.24	0.30

Test 32, Figure 2

Successive concentrates were removed at 1, 3, 6, 10, 15 and 20 minutes of flotation time. The cumulative recoveries of gold and silver and the cumulative concentrate grades have been plotted. A deliberate attempt was made in this test to reduce the amount of flotation reagents.

FIGURE 2 - Results of Test 32



Reagents	301	0.05	-	0.05	0.05	-	0.05
lb/ton	Z-6	-	-	0.05	0.05	-	0.05
Feed	DF-250	0.06	-	0.02	0.02	0.02	0.04

All these tests 24 - 32 were done at the natural pH of the ore (6.6).

Sep. 24 Mineralogical report by Owens entitled: "Mineralogical Investigation of a Sample of Silver-Gold ore from Mount Nansen Mines Ltd., Yukon Territory" has been received.

Sep. 25 Progress Report No. 4

Report on two tests, Test No. 33 and 34. Both tests investigated the use of NaCN and CaO in the primary grind and the rougher flotation circuit, thus providing a longer contact time. Test 33: 55% -200 mesh grinding.

The filtration from all the flotation products were combined and 20 assay tons of it assayed; only trace amounts of gold and silver were observed. Sample of the rougher flotation tailing and of the regrind rougher flotation tailing were cyanided for 24 hours.

The rougher concentrate in Test 33 was reground and then again floated and 3 times cleaned yielding 3 cleaner tailings and a final concentrate. The float and the reground rougher tailing was separately cyanided, each for 24 hours in a solution strength of 1 lb NaCN/t and 1 lb CaO/t. From the final concentrate and the 3 cleaner tailings the following concentration and recoveries were obtained: 6.8; 83.3% Au and 89% Ag. The final concentrate having a concentration ratio of 11.7 and a recovery of 60% Au and 85.6% Ag. Cyanidation of the float tailings showed a 40% recovery for Au and a 55.7% recovery for Ag. Cyanidation of the regrind rougher float tailings showed a 43.6% recovery for Au and 76.8% recovery for Ag.

It has been pointed out that by returning the regrind rougher tailing to the flotation circuit rather than to cyanidation the overall recovery would be somewhat higher.

A 28000 gr test (Test 34) showed similar recoveries. Roasting tests have been planed on the final concentrates; results, however, were not submitted.

FIGURE 3 PROPOSED FLOW SHEET

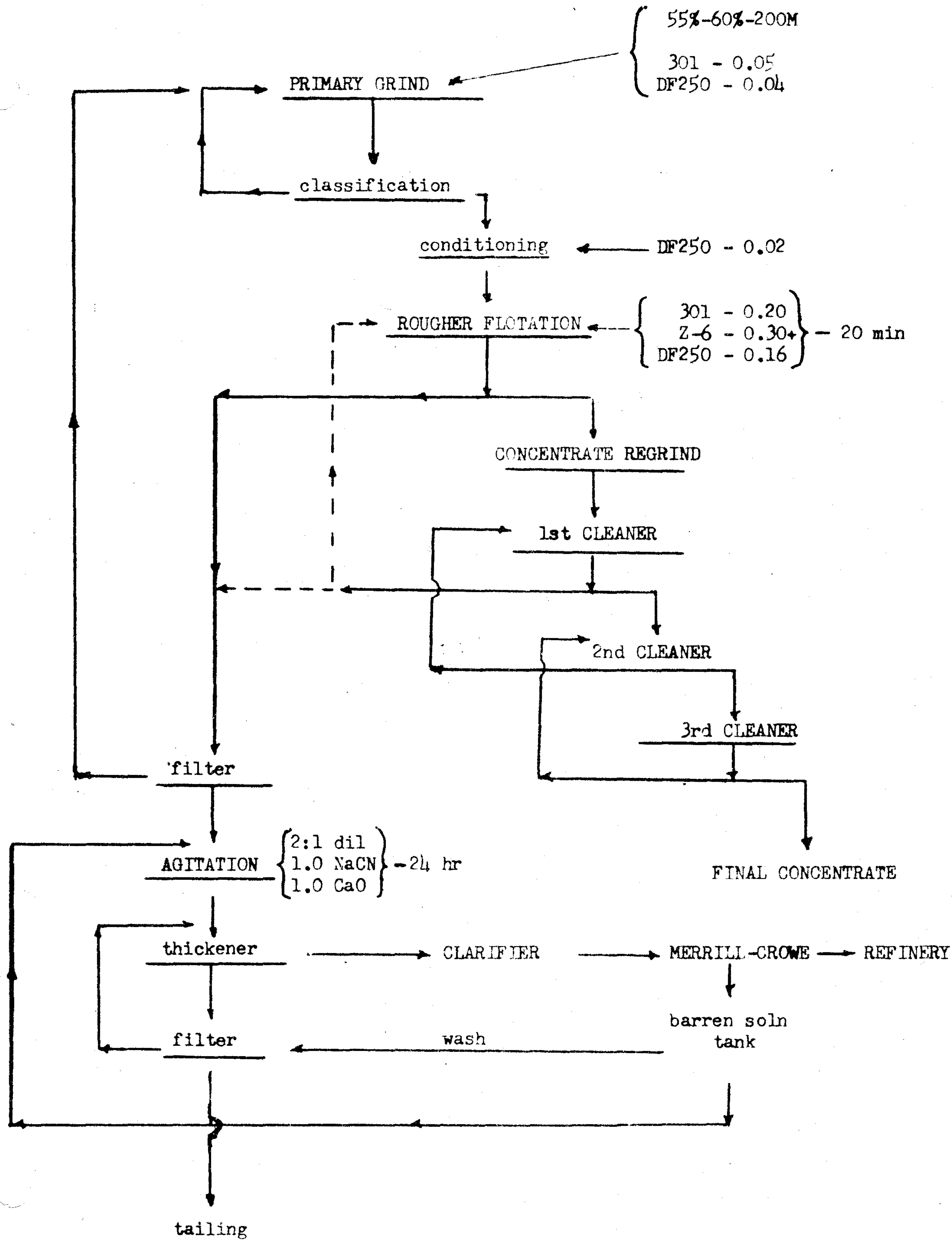


Figure 3: Proposed flow sheet obtained from tests 33/34

M E T A L L U R G I C A L   T E S T S

ON

WEBBER ORE

Author	Ottawa	Ottawa	Britton		Britton	
Date	Nov. 4, 1965	Nov. 4, 1965	Oct. 30, 1964		Aug. 2, 1965	
Test No.	1	8	A	B		
Locality	Webber	Webber			Webber, bulk sample	
Ore-Type	Oxide	Oxide			Oxide	
Head Assay (oz./ton)	0.42 Au, 29.58 Ag	0.42 Au, 29.58 Ag	0.79 Au 51.5 Ag		0.445 Au 32.6 Ag	
Process	Amalgamation	Straight cyanidation	Straight cyanidation		Cyanidation after aeration	
Stage						
Cyanid. time (hours)		24	100	100	72	72
Grind (mesh)	65% -200	77,6% -250	72% -200	96% -200	63% -200	94% -200
Ratio Ore/Conc.						
Recovery Gold (%)	1.0	82.6	82.9	83.1	72.8	74.7
Recovery Silver (%)		84.2	87.4	89.1	83.7	85.4
Total Recovery Gold (%)						
Total Recovery Silver (%)						
Reagent Consumption in Cyanide Process						
NaCN lb/ton ore		6.76	11.4	12.5		
CaO lb/ton ore		13.60	12.6	12.4		
Remarks		Best test of 10 cyanide tests. Increase of contact time by 24 h could increse Ag rec.by app.4%			In presence of lime	

Author	Britton					Britton			Britton		
Date	Sept. 25, 1965					Dec. 29, 1967			Dec. 29, 1967		
Test No.						152-C2			152-2		
Locality	Webber, bulk sample					Webber, 67B			Webber, 67B		
Ore-Type	Oxide					"oxide"			"oxide"		
Head Assay (oz./ton)	0.445 Au 32.6 Ag					0.42 Au 34.5 Ag			0.42 Au 34.5 Ag		
Process	Cyanidation	Cyanid.	Recyanid.	Flot.of pulps	Total	Jig	Cyanid.	Total	Selective Flotation		
Stage									1)Ag-Pb	2)Au	1)+2)
Cyanid. time (hours)	72	72	24				72				
Grind (mesh)	?	?	?			75%-200			71% -200		
Ratio Ore/Conc.				?	?	35		35	10.2	12.7	5.65
Recovery Gold (%)	82.0	81.3	0.8	0.5		2.1	46.9		34.7	32.6	
Recovery Silver (%)	87.1	86.9	1.1	1.7		0.9	48.3		69.0	10.7	
Total Recovery Gold (%)					82.6			49.0			67.3
Total Recovery Silver (%)					89.7			49.2			79.7
Reagent Consumption in Cyanide Process											
NaCN lb/ton ore	8.2	ND	ND				7.8				
CaO lb/ton ore	-	ND	ND				18.1				
Remarks	Na-Hydroxide instead CaO			Cleaned concent.							

Author	Britton				Ottawa				Britton		
Date	Dec. 29, 1967				July 4, 1967				Nov. 9, 1964		
Test No.	152-4				27				-		
Locality	Webber, 67B				Webber				Webber		
Ore-Type	"oxide"				Oxide				Oxide		
Head Assay (oz./ton)	0.42 Au 34.5 Ag				0.37 Au 38.01 Ag				0.79 Au 51.5 Ag		
Process	Selective Flotation			Total	Jig	Table	Flotation	Total	Flotation	Cyanid. of tailings	Total
Stage	1) Ag-Pb	2) Au	3) oxides								
Cyanid. time (hours)										84	
Grind (mesh)	66% -200				100%-10	64%-200	64%-200		63% -200		
Ratio Ore/Conc.	7.4	24.8	9.3	3.8	71	100	13	9.9	11.24		11.24
Recovery Gold (%)	69.9	5.6	10.4		2.0	3.2	29.9		30.9	71.1	
Recovery Silver (%)	80.3	2.9	8.1		9.8	13.5	57.3		76.3	73.0	
Total Recovery Gold (%)				85.9				35.1			80.0
Total Recovery Silver (%)				91.3				80.6			93.6
Reagent Consumption in Cyanide Process											
NaCN lb/ton ore										2.8	
CaO lb/ton ore										10.0	
Remarks	3 conc.	1 conc.	3 conc.						Heads much too high		

Author	Ottawa			
Date	July 4, 1967			
Test No.	29-G			
Locality	Webber			
Ore-Type	Oxide			
Head Assay (oz./ton)	.35 Au, 26.27 Ag			
Process	Cyanidation and Flotation			
Stage	1) Cyanidation	2) Flotation of cyanide tailings	1)+ 2)	
Cyanid. time (hours)	24			
Grind (mesh)	64.0% -200	64.0% -200		
Ratio Ore/Conc.		9.5	9.5	
Recovery Gold (%)	72.2	18.1		
Recovery Silver (%)	80.0	29.4		
Total Recovery Gold (%)			77.2	
Total Recovery Silver (%)			86.1	
Reagent Consumption in Cyanide Process				
NaCN lb/ton ore				
CaO lb/ton ore				
Remarks	Tests are only partially successful as only an additional 5.0% of the Au and 5.8% of the Ag were recovered in the best test.			

M E T A L L U R G I C A L T E S T S

ON

HUESTIS ORE

Author Date Test No.	Britton Feb. 23, 1966 B.C. 1	Britton Feb. 23, 1966 B.C. 2	Britton Dec. 29, 1967 152-C1			Britton Feb. 23, 1966 H 1			
Locality Ore-Type Head Assay (oz./ton)	Huestis Sulphide 0.70 Au 15.8 Ag	Huestis Sulphide 0.70 Au 15.8 Ag	Huestis, 67A Sulphide 0.5 Au 29.1 Ag			Huestis Sulphide 0.70 Au 15.8 Ag			
Process	Cyanidation	Cyanidation	1) Jig	2) Cyanid.	1)+ 2)	Flotation			
Stage						1) Ag-Pb	2) Pyr-Apy	1)+ 2)	
Cyanid. time (hours)	72	144		72					
Grind (mesh)	80% -200	99% -200	68%-200			70%-200			
Ratio Ore/Conc.			50		50	8.3	5.9	3.5	
Recovery Gold (%)	20	24.7	8.2	8.4		32	60		
Recovery Silver (%)	27.5	30.0	0.9	22.2		90	7		
Total Recovery Gold (%)					16.6			92	
Total Recovery Silver (%)					23.1			97	
Reagent Consumption in Cyanide Process									
NaCN lb/ton ore	3.04	4.64	2.9						
CaO lb/ton ore	14.4	16.4	17.0						
Remarks						Concentrates not cleaned.			

Author	Britton					Britton	Britton	
Date	Feb. 23, 1966					Feb. 23, 1966	Feb. 23, 1966	
Test No.	H 2					B.C. 3	B.C. 4	
Locality	Huestis					Huestis	Huestis	
Ore-Type	Sulphide					Conc. 2) of H2	Conc. 2) of H2	
Head Assay (oz./ton)	0.70 Au 15.8 Ag					2.64 Au 2.9 Ag	2.64 Au 2.9 Ag	
Process	Flotation					Cyanidation	Cyanidation after Roasting	H2 + B.C. 4
Stage	1) Ag-Pb		2) Py-Apy		1) + 2)			
Cyanid. time (hours)						72 tot.	72 (tot.)	
Grind (mesh)	71%-200					99% -200	76% -325	
Ratio Ore/Conc.	19.7	11.2	8.25	25	$3.3 < x > 5.8$			$11.2 < x > 19.7$
Recovery Gold (%)	21.5	20.1	46.1	2.9		3.6	76.1	
Recovery Silver (%)	83.6	10.5	2.2	0.8		50.7	54.3	
Total Recovery Gold (%)					90.6			78.9
Total Recovery Silver (%)					97.1			95.7
Reagent Consumption in Cyanide Process								
NaCN lb/ton ore						6.8/t conc.	5.2/t conc.	
CaO lb/ton ore						27.6/t conc.	26.8/t conc.	
Remarks	Final conc.	Clnr. Tail.	Final Conc.	Clnr. Tail.				

Author	Britton			Britton		Britton		
Date	Feb. 23, 1966			March 14, 1966		Dec. 9, 1966		
Test No.	H 3			H 4		H 5		
Locality	Huestis			Huestis		Huestis		
Ore-Type	Sulphide			Sulphide		Sulphide		
Head Assay (oz./ton)	0.70 Au 15.8 Ag			0.70 Au 15.8 Ag		0.70 Au 15.8 Ag		
Process	Flotation			Flotation		Flotation		
Stage	Ag-Pb			Ag-Pb		1) Ag-Pb	2) Au	1)+ 2)
Cyanid. time (hours)								
Grind (mesh)	81% -200			70% -200 and reground of +325 rghr. conc.		70% -200		
Ratio Ore/Conc.	27.4	6.9	$6.9 < x > 27.4$	38	8.6	6	5.25	2.65
Recovery Gold (%)	12.6	37.6		7.6	28.6	45.7	49.8	
Recovery Silver (%)	77.0	94.5		70.8	93.0			
Total Recovery Gold (%)			37.6					95.5
Total Recovery Silver (%)			94.5					
Reagent Consumption in Cyanide Process								
NaCN lb/ton ore								
CaO lb/ton ore								
Remarks	Final Conc.	Rough Conc.	Total	Final Conc.	Rough Conc.	1 Rough. Conc.	11 Rough. Conc.	

Author	Britton				Britton			Britton	Britton
Date	Dec. 29, 1967				March 18, 1968			March 18,1968	March 18, 1968
Test No.	152-1				152-7				
Locality	Huestis, 67A				Huestis			Huestis	Huestis
Ore-Type	Sulphide				Sulphide			Sulphide	Sulphide
Head Assay (oz./ton)	0.5 Au 29.1 Ag				0.5 Au 29.1 Ag				
Process	1) Jig	Flotation		1)+2)+3)	Flotation			Roasting	Leaching
Stage		2) Ag-Pb	3) Au		Bulk				
Cyanid. time (hours)									
Grind (mesh)		63% -200			78% -200				
Ratio Ore/Conc.	38.5	20	5.4	4.2	5.4	3.5	3.5 < x > 5.4		
Recovery Gold (%)	8.4	11.4	74.5		90.6	92.0			
Recovery Silver (%)	0.8	86.2	9.1		95.5	96.6			
Total Recovery Gold (%)				94.3			92.0		
Total Recovery Silver (%)				96.1			96.6		
Reagent Consumption in Cyanide Process									
NaCN lb/ton ore									
CaO lb/ton ore									
Remarks		3 conc.	3 conc.		Final Conc.	Rough Conc.		See Summary Review	See Summary Review

M E T A L L U R G I C A L   T E S T S

ON

HUESTIS & WEBBER ORE

Author	Ottawa	Ottawa	Ottawa
Date	May 6, 1968	May 6, 1968	May 6, 1968
Test No.	7	10	9
Locality	Huestis and Webber	Huestis and Webber	Huestis and Webber
Ore-Type	40% oxide, 60% sulfide	40% oxide, 60% sulfide	40% oxide, 60% sulfide
Head Assay (oz./ton)	.39 Au, 17.01 Ag	0.38 Au, 16.72 Ag	0.40 Au, 16.94 Ag
Process	Flotation	Flotation	Flotation
Stage			
Cyanid. time (hours)			
Grind (mesh)	70% -200	90% -200	90% -200
Ratio Ore/Conc.	3.9	2.1	2.5
Recovery Gold (%)			
Recovery Silver (%)			
Total Recovery Gold (%)	81.1	83.1	83.6
Total Recovery Silver (%)	90.6	94.0	93.8
Reagent Consumption in Cyanide Process			
NaCN lb/ton ore			
CaO lb/ton ore			
Remarks	No cleaning of the rougher concentrate	No cleaning of rougher concentrate. 7 flotation concentrate were removed.	No cleaning of the rougher concentrate.

Author	Ottawa			Britton		Britton		
Date	June 18, 1968			Dec. 29, 1967		Dec. 29, 1967		
Test No.	14			152-3		152-6		
Locality	Huestis and Webber			Huestis and Webber		Huestis and Webber		
Ore-Type	40% oxide, 60% sulfide			Sulfide and Oxide		Sulfide and Oxide		
Head Assay (oz./ton)	0.34 Au, 15.98 Ag			0.46 Au 31.8 Ag		0.46 Au 31.8 Ag		
Process	Flotation			Flotation		Flotation Bulk		
Stage	1) rougher flot. conc.	2) scavanger conc.	1)+2)					
Cyanid. time (hours)								
Grind (mesh)	75% -200	75% -200		64% -200		64% -200		
Ratio Ore/Conc.	4	12.2	3.0	9.15	5.3	6.6	4	4<x>6.6
Recovery Gold (%)	76.0	5.3		63.4	77.3	77.9	85	
Recovery Silver (%)	90.8	2.6		77.5	87.2	82.3	91	
Total Recovery Gold (%)			81.3					85
Total Recovery Silver (%)			93.4					91
Reagent Consumption in Cyanide Process								
NaCN lb/ton ore								
CaO lb/ton ore								
Remarks				Final conc.	Rough + scav. conc.	Final conc.	Rough conc.	

Author	Britton						Ottawa			
Date	Dec. 29, 1967						June 18, 1969			
Test No.	152-5						18			
Locality	Huestis and Webber						Huestis and Webber			
Ore-Type	Sulfide and Oxide						40% oxide, 60% sulfide			
Head Assay (oz./ton)	0.46 Au 31.8 Ag						0.39 Au, 17.02 Ag			
Process	Flotation					Cyanid. of ox.conc.	Flotation and Cyanidation			
Stage	1) Sulfides		2) Oxides		1) + 2)		1) Flotation		2) Cyanidation of flot.tail.	1)+2)
Cyanid. time (hours)						72			24	
Grind (mesh)	64% -200						75%-200		75% -200	75%-200
Ratio Ore/Conc.	6.1	3.9	60	13.7	3 < x > 5.6		3.1	10.5		3./<x>10.5
Recovery Gold (%)	82.8	89.7	1.6	3.8		53	80.1	64.3	65.2	
Recovery Silver (%)	86.4	91.8	1.6	3.3			90.1	81.4	63.1	
Total Recovery Gold (%)					93.5					93.1
Total Recovery Silver (%)					95.1					96.3
Reagent Consumption in Cyanide Process										
NaCN lb/ton ore						16.3			0.56	
CaO lb/ton ore						27.5			2.40	
Remarks	Final conc.	Rough conc.	Final conc.	Rough conc.			Rough.conc. uncleaned	Final conc.	No improve.by finer grinding or longer cyanidation	

Author	Ottawa				Ottawa				
Date	July 25, 1968				Sept. 25, 1968				
Test No.	29				33				
Locality	Webber and Huestis				Webber and Huestis				
Ore-Type	40% oxide, 60% sulfide				40% oxide, 60% sulfide				
Head Assay (oz./ton)	0.39 Au, 16.83 Ag				0.42 Au 15.15 Ag				
Process	Flotation and Cyanidation				Flotation and Cyanidation				
Stage	1) Flotation		2) Cyanidation of float.tail.	1) + 2)	1) Flotation		2) Cyanidation		1) + 2)
Cyanid. time (hours)			24				24		
Grind (mesh)	55-60% -200	55-60% -200	55-60% -200		55% -200		55% -200		
Ratio Ore/Conc.	3.6	12.2		3.6<x>12.2	7.5	11.7			7.5 < x >11.7
Recovery Gold (%)	83.5	59.8	66.7		83.3	60.4	40.0	43.6	
Recovery Silver (%)	92.2	75.2	56.4		89.0	83.8	55.7	76.8	
Total Recovery Gold (%)				94.5					90.3
Total Recovery Silver (%)				96.6					96.1
Reagent Consumption in Cyanide Process									
NaCN lb/ton ore									
CaO lb/ton ore									
Remarks	Rougher conc. regrind of rougher concentrate	final conc.	Solution strength 1 lb NaCN/t 1 lb CaO/t		Rougher conc.	Final conc.	Regrind float tail.	Regrind rougher float tailings	
					If the regrind rough. tail. were returned to flotation recovery should be better.				

File: Mt. Nansen Mines Limited

Subject: Metallurgical testing

November 5, 1969
Dr. F. Bianconi

SUMMARY OF THE METALLURGICAL TESTS

PERFORMED ON MOUNT NANSEN ORE

by

COMINCO LIMITED

1968

S. Gray required two tests at grinds of 65 and 85% -200 mesh to determine whether or not the finer grind would substantially lower the gold and silver losses in the flotation tailings.

Head Assay: 0.40 oz/ton Au and 15.9 oz/ton Ag. (Composite of Huestis and Webber)

Test 1

Grinding: 56.3% -200 mesh

Results:

	% Wt.	Au oz/ton	Ag oz/ton	% Distribution Au	% Distribution Ag
Sulphide Cleaner Concentrate	10.15	2.60	126.5	68.0	79.5
Sulphide Cleaner Tailing	2.49	0.58	18.4	3.6	2.8
Oxide Recleaner Concentrate	3.00	0.6	33.3	4.6	6.1
Oxide Recleaner Tailing	4.11	0.32	7.5	3.4	1.9
Oxide Cleaner Tailing	7.31	0.18	3.6	3.4	1.6
Oxide Rougher Tailing	72.94	0.09	1.8	17.0	8.1
Calc. Feed	100.0	0.39	16.18	100.0	100.0

Test 2

Grinding: 81.8% -200 mesh

Results:

	% Wt.	Au oz/ton	Ag oz/ton	% Distribution Au	% Distribution Ag
Sulphide Cleaner Concentrate	9.1	2.80	138.0	70.7	78.3
Sulphide Cleaner Tailing	4.6	0.36	17.5	4.7	5.0
Oxide Recleaner Concentrate	2.3	0.52	38.5	3.3	5.5
Oxide Recleaner Tailing	4.5	0.16	4.9	1.9	1.4
Oxide Cleaner Tailing	10.1	0.14	3.2	3.9	2.0
Oxide Recleaner Tailing	69.4	0.08	1.8	15.5	7.8
Feed	100.0	0.36	16.04	100.0	100.0

Conclusion

"The fine grind did not give a substantially lower tailing but resulted in a higher grade sulphide concentrate".

Cleaned sulphide and oxide concentrates:

Test 1
72.6% Au
85.6% Ag
Concentration ratio:7.6/1

Test 2
74.0% Au
83.8% Ag
Concentration ratio:8.8/1

<u>File</u>:	Mount Nansen Mines Limited	December 4, 1969 Dr. F. Bianconi Dr. R. Saager
<u>Subject</u>:	Metallurgical testing	

SUMMARY AND CONCLUSION

Mineralogical and metallurgical investigations revealed clearly the presence of two ore types in the known workings of the Mount Nansen property. The ore in the Huestis section is a complex <u>sulfide ore</u>, arsenopyrite, pyrite, sphalerite,' jamesonite, galena and freibergite being the main constituents. The ore in the Webber section is primarily on <u>oxide ore</u>. Its constituents are rather difficult to determine. However, they seem to be a complex assemblage of bindheimite, anglesite, and various other alteration products of the sulfides present in the Huestis ore. Geological implications give rise to the assumption that with increased depth the Webber ore will gradually change its nature into a sulfide ore. The mineralogical investigation carried out by Hazen indicates that the primary Webber sulfide ore differs from the Huestis sulfide ore inasmuch as it might carry higher Ag and Sb contents.

The present ratio between sulfide and oxide ore is approximately 6 : 4. Naturally, this ratio is subject to changes in the course of mining, however, one never can expect to mine exclusively sulfides as long as mining operations are carried out in shallow depths. This implication is confirmed by the presence of oxidized ore in all the trenches and by the presence of highly oxidized ore in the 4100 level at Brown McDade.

Metallurgical tests carried out separately on the two ore types indicate a good recovery and concentration for both the sulfide and the oxide ore.

1) Huestis ore.

The best recoveries and concentration of this ore has been achieved by making use of a jig followed by selective flotation of a "silver - lead" concentrate and a "gold" concentrate (Test 152 - 1 by Britton).

	Jig.	Ag-Pb	Au	Total
conc. ratio	38.5	20	5.4	4.2
rec. Au	8.4%	11.4%	74.5%	94.3%
rec. Ag	0.8%	86.2%	9.1%	96.1%

The recovery is good but the concentration ratio is rather poor due to the fact that the "gold" concentrate is of low grade. Roasting tests performed on other bulk concentrates indicate an approximately 25% reduction of the weight, an 80 - 90% reduction of the As content and a 50% reduction of the Sb content. The gold loss and silver loss experienced was 2.5 - 3,5% and 0.2 - 1.6% respectively (March 18, 1968; Britton).

These tests indicate that such an approach followed by cyanidation of the roasted "gold" concentrate would yield both a high recovery and a high concentration for the sulfide ore.

A flotation of a bulk concentrate yielded. (Test 151 - 7 by Britton):

conc. Ratio	5.4
rec. Au	90.6%
rec. Ag	95.5%

The cyanidation of the tailings seems to be uneconomical.

2) Webber ore.

Best results were obtained by straight cyanidation of the oxide ore (Test 8, Ottawa)

conc. ratio	---	
rec. Au	82.6%	consumption NaCN 6.8 lbs/ton ore
rec. Ag	84.2%	consumption CaO 13.6 lbs/ton ore
Ag. time	24 hrs.	

Gravitative concentration followed by flotation yielded a poor recovery and a moderate concentration ratio. (Test 27, Ottawa)

	Jig	Table	Flot.	Total
conc.ratio	71	100	13	9.9
rec. Au	2.0%	3.2%	29.9%	35.1%
rec. Ag	9.8%	13.5%	57.3%	80.6%

Flotation followed by cyanidation of tailings indicate a reasonable recovery and concentration (Britton Nov. 9, 1964)

	Flot.	Cyanid of tailings	Total
conc.ratio	11.24	---	11.2
rec. Au	30.9%	71.1%	80.0%
rec. Ag	76.3%	73.0%	93.6%

NaCN consumption	2.8 lbs/ton ore
CaO consumption	10.0 lbs/ton ore

The grade of the heads used in this test was much too high.

The above tests indicate that economically the best results for the Webber ore can be achieved by straight cyanidation.

3) Huestis and Webber.

Composite samples of the sulfide and oxide ore have been tested excessively both by Britton, using a 1 : 1 mixture and by Ottawa using a 6 : 4 mixture. Several approaches have been performed and the following "best" results were obtained.

Flotation of a bulk concentrate. (Test 152 - 6, Britton)

conc.ratio	4 - 6.6
rec. Au	85%.
rec. Ag	91%.

The Webber ore portion used for this test was not representative having a too low degree of oxidation. The recoveries obtained in this test must therefore be considered to be rather optimistic.

Flotation of a sulfide and oxide concentrate.
(Test 152 - 5, Britton, this test has been used as a base for the feasibility study of Campbell).

	Sulfides	Oxides	Total
conc.ratio	3.9 - 6.1	13.7 - 60	3 - 5.6
rec.Au	89.7 - 82.8%	3.8 - 1.6%	93.5 - 84.4%
rec. Ag	91.8 - 86.4%	3.3 - 1.6%	95.1 - 88.0%

The same considerations as for test 152 - 6 must be made in this test. This is the reason why practically all of the gold and silver is recovered in the sulfide concentrate.

Flotation of a bulk concentrate followed by cyanidation of flotation tailings (Test 33, Ottawa).

	Flotation	Cyanidation of Tailings	Total
conc.	7.5 - 11.7		7.5 - 11.7
rec. Au	83.3 - 60.4%	43.6%	77.7 - 90.6%
rec. Ag	89.0 - 83.8%	76.8%	96.2 - 97.4%

This is the best test carried out on composite samples. By using this approach most of the metals recovered in the flotation derive from the Huestis sulfide fraction and most of the metals recovered in the cyanidation derive from the Webber oxide fraction.

From the discussion of the above tests it seems desirable that the oxide and sulfide ores are treated separately. The treatment of a composite ore has the disadvantage that in the cyanide stage a large volume must be handled. More than 50% of this volume is derived from the sulfide fraction and contains very limited amounts of gold and silver which economically do not justify a cyanide treatment. This conclusion is solely based on the presently available metallurgical data and does not comprise any operational considerations.

<u>File:</u>	Mount Nansen Mines Limited	December 17, 1969
<u>Subject:</u>	Metallurgy	Dr. F. Bianconi

THE MOUNT NANSEN MILLING OPERATION

September 23, 1968 to April 8, 1969

The Mount Nansen mill (flotation plant) was in operation during six months, from September 23, 1968 to April 8, 1969 for a total of 191 operating days. An attempt to compile a metallurgical balance for this period is given here.

The compilation shown in the attached table is based on the daily mill reports of the Mill-Superintendent, Mr. C. Coffey. The table contains all data pertaining to the mill operation, that is, tonnages of heads, of concentrate and of tailings, ratio ore/concentrate, gold - silver assays and contents and the recoveries.

However, this compilation is of little value and has to be analysed critically since most of the data are calculated. Only the concentrate and tailing grade and the concentrate tonnage are observed values. Since most of the values were calculated the metallurgical balance is perfect, but there are no possibilities to control its validity.

It is attempted to explain the calculations of the metallurgical balance by using one example, choosen at random from the daily mill reports (Jan. 12, 1969).

<u>Date - January 12th, 1969</u>

SDT Reported Milled - 189

SDT Concentrate Produced - 12.1005

Assay	Au Oz/Ton	Au Oz/Ton
Heads	.420	10.140
Conc.	2.571	76.458
Tails	.100	1.700

R/C Au		7.72
R/C Ag		8.86
SDT of feed indicated - Au		93
SDT of feed indicated - Ag		107
	Average	$\frac{200}{2}$

Report 100 SDT

Metallurgical Balance

<u>Au</u>	SDT	Assay Au	Oz Au	%Recovery	R/C
Heads	100	.399	39.900	100	
Conc.	12.1005	2.571	31.110	77.95	8.3/1
Tails	87.8995	.100	8.790	22.05	

<u>Ag</u>	SDT	Assay Ag	Oz Ag	%Recovery	R/C
Heads	100	10.74	1074.611	100	
Conc.	12.1005	76.458	925.182	86.09	8.3/1
Tails	87.8995	1.700	149.429	13.91	

-/2

1) SDT (short dry tons) Reported Milled - 189 tons. Since there was was no weightometer installed at the mill, the mill-feed was estimated at the conveyor belts, which connect the fine ore bins and the ball mills. 1 foot of ground ore was taken from the conveyor periodically, and weighted on a oldish balance. The speed of the conveyor belt was estimated and the mill-feed then calculated from the two values. This value of course, is affected by a number or errors, i.e. variations of the ore - volume on the conveyor belt, variation in the speed of the conveyor belt, etc.

2) SDT (short dry tons) concentrate produced - 12.1005 tons. This value can be considered as fairly correct, since it was obtained by direct weighting on a modern scale. The moisture content was determined in the assay labor and subtracted from the obtained weight.

3) Assay values: these are the assays as reported by the resident assayer. It is difficult to estimate the error of the assays: the only check available were the check assays done by the Whitehorse Assay Office. The comparison of the results supplied by the mine assayer and by Whitehorse shows a good agreement in the higher grades, but larger discrepancies in the low grades.

A calculation of the metallurgical balance, based on the above values does not tally; see following example:

SDT reported milled: 189 tons
SDT concentrate produced: 12.1005 tons
Ratio ore: concentrate: 15.6192
Tailings: 176.8995 tons

<u>Gold balance</u>

Heads		0,420 oz/ton	(assayed)
(a)	or	79.38 oz	(oz/ton x tons feed)
Concentrate		2.571 oz/ton	(assayed)
(b)	or	31.1104 oz	(oz/ton x tons concentrate)
Recovery		39.19%	
Tailings		0.100 oz/ton	(assayed)
(c)	or	<u>17.69 oz</u>	(oz/ton x tons tailings)

(c) is also equal (a) - (b), in this case:
(c) = 79.38 - 31.1104 = <u>48.2696 oz</u>,
which is almost three times larger than the other figure.

<u>Silver balance</u>

Heads		10.14 oz/ton	(assayed)
(a)	or	1916.46 oz	(assay x tons feed)
Concentrate		76.458 oz/ton	(assayed)
(b)	or	925.18 oz	(assay x tons concentrate)
Recovery:		48.28%	
Tailings		1.7 oz/ton	(assayed)
(c)	or	<u>300.73 oz</u>	(assay x tons tailing)

or (c) = (a) - (b) = 1916.46 - 925.18 = <u>991.28 oz</u>
Here the difference is even larger.

Cy Coffey has discovered these large discrepancies very soon during the operation and attributed them to a wrong mill-feed tonnage. During October 1968, the discrepancies were sometimes so large, that the calculated recoveries are higher than 100%! Starting December 1, 1968, Coffey introduced a new calculation of the metallurgical balance, based on the assays and on the weight of the concentrate only.

His calculation of the metallurgical balance is as follows:

1) Concentration ratio calculated from the assays:

If	a=	Grade of heads	(known)
	b=	Grade of concentrate	(known)
	c=	Grade of tailings	(known)
	x=	Tonnage of concentrate	(known)
	y=	Tonnage of tailings	(unknown)
	x+y=	Tonnage of heads	(unknown)

Then

$$(x+y)\ .a = bx + cy$$
$$ax + ay = bx + cy$$
$$x.(b-a) = y.\ (a - c)$$
$$y = x.\ \frac{(b - a)}{(a - c)}$$

Concentration ratio is

$$\frac{x + y}{x} = \frac{x + \frac{x.(b - a)}{(a - c)}}{x}$$
$$= 1 + \frac{b - a}{a - c} =$$
$$= \frac{a - c + b - a}{a - c} = \frac{b - c}{a - c}$$

$$\text{Concentration ratio} = \frac{\text{conc. grade - tailing grade}}{\text{heads grade - tailing grade}}$$

In our example it results:

R/c Au (concentration ratio indicated by Au) = 7.72
R/c Ag (concentration ratio indicated by Ag) = 8.86

And:

SDT of feed indicated by Au = R/c Au x tons concentration = 93 tons
SDT of feed indicated by Ag = R/c Ag x tons concentration =107 tons

$$\text{Average} = \frac{93 + 107}{2}\ \text{tons} = 100\ \text{tons}$$

From this calculated mill-feed, Cy Coffey calculated the grade of the heads solving the equation (x + y) .a = bx = cy after a.

$$a = \text{head assay} = \frac{bx + cy}{x + y}$$

In our example:

calc. head assay Au = 0.399 oz/ton (assayed: 0.42)
calc. head assay Ag = 10.74 oz/ton (assayed: 10.14)

From these calculated figures we obtain:

Concentration ratio from calculated feed = 8.3
(concentration ratio from estimated feed = 15.6)
Recovery Au from calculated feed = 77.95%
(Recovery Au from estimated feed = 39.19%)
Recovery Ag from calculated feed = 86.09%
(Recovery Ag from estimated feed = 48.28%)

In other words: Cy Coffey, not having a weightometer in the mill, was forced to calculate his heads (tons and grade) back from the best reliable data he had at disposal, namely concentrate weight and assays of heads, concentrate and tailings. The error of the calculated mill feed is directly affected by the errors of the assays. The total mill-feed over the six months of production is as follows:

i) from estimates at the conveyor belt	22,633 tons
ii) from the calculations	18,017 tons
iii) from the engineer's reports (ore from mine and stockpile)	20,431 tons

i) Was already discussed and can be considered of no value,

ii) is used in the metallurgical summary for the operation and is probably too low,

iii) was also estimated, since there is no weightometer in order to weight the loaded cars. The tonnage factor was also only estimated and was never determined. The engineer reported 69% tons coming from the stockpile and 13,511 tons coming from the stopes. Here again there is an other discrepancy, since the engineer reports 14,224 tons pulled from the stopes. The conclusion of these considerations is that the actual total tonnage of mill feed lies probably somewhere between 18,000 and 21,000 tons, and that the total mill-feed of 18,017 tons as given in the summary table represents probably a minimum.

If this is correct, then the reported average concentration ratio of 12.2 has to be considered as a minimum and the reported average recoveries of 60.5% for gold and of 79.1% for silver have to be considered as maxima.

SUMMARY OF MOUNT NANSEN MILLING OPERATION

(SEPTEMBER 23, 1968 to APRIL 8, 1969)

	Days operating	Tons ore milled (calc.)	Aver. tons/day (calc.)	Concentrate		Av, Ratio ore:conc. (calc.)	Tailings tons (calc.)	GOLD BALANCE							SILVER BALANCE						
				tons (obs.)	tons/day (calc.)			Heads		Concentrate		Recovery	Tailings		Heads		Concentrate		Recovery	Tailings	
								oz/ton (calc.)	oz Au (calc.)	oz/ton (obs.)	oz Au (calc.)	Au % (calc.)	oz/ton (obs.)	oz Au (calc.)	oz/ton (calc.)	oz Ag (calc.)	oz/ton (obs.)	oz Ag (calc.)	Ag % (calc.)	oz/ton (obs.)	oz Ag (calc.)
1968 SEPT.	8	390	48.7	18.9780	2.37	20.5	371	0.121	47.200	0.878	16.670	35.3	0.082	30.530	2.372	925.300	16.543	313.969	33.9	1.648	611.331
OCT.	24	2,265	94.4	184.0903	7.67	12.3	2,081	0.256	580.866	1.409	259.356	44.6	0.154	321.510	5.100	11,549.790	45.664	8,406.342	72.8	1.510	3,143.448
NOV.	30	3,967	132.2	226.7940	7.56	17.5	3,740	0.230	916.595	1.730	392.457	42.8	0.140	524.048	5.015	19,897.868	50.545	11,465.453	57.6	2.255	8,432.415
DEC.	31	2,353	75.9	175.0210	5.65	13.4	2,178	0.224	526.140	2.024	354.265	67.3	0.078	170.975	4.442	10,453.100	53.921	9,437.392	90.3	0.558	1,215.372
1969 JAN.	31	3,270	105.5	273.3980	8.82	12.0	2,996	0.268	876.724	2.269	620.304	70.7	0.085	256.410	6.570	21,489.930	65.234	17,834.915	83.0	1.219	3,653.018
FEB.	28	4,083	145.8	436.1000	15.57	9.4	3,647	0.315	1,284.937	2.112	921.094	71.7	0.099	363.861	7.887	32,201.977	63.455	27,672.778	85.9	1.242	4,529.299
MARCH	31	1,343	43.3	128.4430	4.14	10.5	1,214	0.273	366.349	1.693	217.463	59.3	0.122	148.754	6.428	8,633.856	61.396	7,885.871	91.3	0.624	757.975
APRIL	8	346	43.3	32.6465	4.08	10.6	313	0.240	84.000	1.583	51.689	61.5	0.103	32.311	6.690	2,312.340	62.447	2,038.697	88.2	0.874	273.643
TOTAL	191	18,017		1,475.4708			16,540		4,682.811		2,833.298			1,848.399		107,464.141		85,055.417			22,616.501
AVERAGE			94.3		7.72	12.2		0.260		1.920		60.5	0.112		5.965		57.646		79.1	1.367	

File: Mount Nansen Mines Limited

March 6, 1970
Dr. F. Bianconi

Subject: Metallurgical testing

COLLECTING OF SAMPLES FOR METALLURGICAL TESTS AT THE MOUNT NANSEN MINE

On February 6, 1970 we were informed by Mr. Mercier on a management decision to collect the samples for the metallurgical tests prior to the shut-down of the present caretaker operation at the mine. I went to the mine on February 12, where I supervised the collection and the proper storage of the samples. The operation was terminated on Feb.27.

The samples were collected following partly the scheme proposed by Hazen Laboratory. They asked for one "small" sample weighing approximately 1 ton, taken from at least 30 points along the vein, and one "large" sample weighing approximately 10 tons, taken in the same manner, from each of the mines, the Huestis and the Webber. Total required: 2 tons of "small" samples and 20 tons of "large" samples.

The attached lists and maps furnish a detailed inventory of all the samples taken and give information on bag number, ore shoot, exact location and weight. Although the lists and maps are self-explanatory, some remarks on the actual technique used for collecting the samples are needed.

A) "Small" Samples

1. Huestis mine. A total of 28 samples were collected in the Huestis mine, 14 from the 4300 level, 14 from the 4100 level. They are stored in separate canvas bags, numbered according to the attached list. They were taken from 12 ore shoots, in both the drifts and the stopes. Since the samples were taken across 4' width, the material should have an average grade close to the one calculated for the Huestis mine, namely 0.42 oz.Au/ton and 12.06 ozAg/ton One "sample", No 29, is of concentrate from the previous milling operation; it was obtained from Huestis ore and assayed 1.87 oz. Au/ton and 77.01 oz.Ag/ton. This concentrate could be used as a "sweetener". The material includes all fractions from the fine up to coarse fragments (8 inches) and is generally dry, with exception of samples 15 to 21, taken from a wet section of the mine.

2. Webber mine. A total of 34 samples were collected in the Webber mine (4260 level) from 12 ore shoots. The sampling was difficult here, since the ground is hardly frozen and since we had no pneumatic hammer, so the sampling had to be done with the scaling bar. 29 samples were taken across 4' width; 5 (no 59 to 63) are handpicked ore, partly of high grade, which can be used as a "sweetener", if necessary to reach the calculated average grade of the Webber Mine, 0,338 oz.Au/ton and 20.28 oz.Ag/ton. The material includes all fractions from the fine up to coarse fragments (8 inches) and is partially wet, to the presence of ice in the drift.

The 63 samples,totalling 4,148 lbs.,are presently stored at Whitehorse (Airport, hangar of Mr. E. Phillips).

It must be noted that the composite sample should assay approximately 0.39 oz.Au/ton and 15.5 oz.Ag/ton. This is the average grade estimated for the present accessible ore reserves.

B) "Large" Samples

In order to obtain the 20 tons of bulk samples within a reasonable time, the samples were taken from only few selected points. These points were drilled and blasted.

1. Huestis. 6 tons were obtained from four points at the 4100 level and 4 tons from two points at the 4300 level. The reason for taking only six sample points is that most of the ore shoots are partly stoped. This condition makes the collection of large samples in the stopes impossible.

2. Webber. The 10 tons were obtained from 5 sampling points. Because of the fact that the samples were taken from only few points, their average grade could be fairly different from the calculated average grade of the ore reserves. Therefore the samples were stored individually, in order to make a blending possible.

The 11 samples, totalling 20 tons, are presently stored at Carmacks, Y.T.

F. Bianconi

FB:vs

Dr. F. Bianconi

HUESTIS: Small samples for metallurgical testing

LEVEL	BAG	ORE SHOOT	LOCATION	WEIGHT (lbs
HUESTIS 4300 LEVEL	1	H43 - 12 - 585	Stope, 72' east of W-Manway	77
	2	H43 - 12 - 585	Stope, 51' east of W-Manway	78
	3	H43 - 12 - 585	Stope, 23' east of W-Manway	65
	4	H43 - 12 - 585	Stope, 20' west of E-Face	82
	5	H43 - 12 - 588	Station H54 + 58' west	71
	6	H43 - 12 - 588	Station H54 + 21' west	73
	7	H43 - 12 - 590	Stope, 50' east of W-Face	74
	8	H43 - 12 - 590	Stope, 43' west of E-Face	61
	9	H43 - 12 - 591	Station 651 + 58' east	58
	10	H43 - 12 - 591	Station 651 + 95' east	69
	11	H43 - 12 - 594	Stope, Chute No.1, broken ore	80
	12	H43 - 12 - 594	Station 613 + 79' east	63
	13	H43 - 13 - 595	Station 645 + 39' west	62
	14	No. 13 Vein	Face, Station H99 + 79' west	68
HUESTIS 4100 LEVEL	15	H41 - 12 - 585	Station H97 + 77' west	78
	16	H41 - 12 - 585	Station H97 + 30' west	74
	17	H41 - 12 - 585	Station H97 + 5' east	83
	18	H41 - 12 - 585	Station H96	74
	19	H41 - 12 - 585	Station H94 + 50' west	66
	20	H41 - 12 - 585	Station 713 + 20' west	87
	21	H41 - 12 - 585	Station 713 + 10' west	96
	22	H41 - 12 - 588	Stope, East Face	64
	23	H41 - 12 - 588	Drift between Chutes No.2 & No.3	63
	24	H41 - 12 - 590	Station D1 - 18 + 7' west	63
	25	H41 - 12 - 591	Station D1 - 16 + 5' west	73
	26	H41 - 12 - 591	Station D1 - 15 + 2' east	67
	27	H41 - 12 - 594	Stope, West Face + 20'	70
	28	H41 - 12 - 594	Stope, East Manway + 5' West	74
	29	Concentrate	Huestis 4100 + Huestis 4300	77
			TOTAL WEIGHT	2,090

WEBBER: Small samples for metallurgical testing

BAG	ORE SHOOT	LOCATION	WEIGHT (lbs)
30	W43 - 2 - 558	Stope, Station 148 + 20' west	69
31	W43 - 2 - 558	Stope, Station 148 + 30' west	56
32	W43 - 2 - 558	Stope, Station 148 + 47' west	59
33	W43 - 2 - 558	Stope, Station 148 + 67' west	77
34	W43 - 2 - 558	Stope, Station 148 + 82' west	82
35	W43 - 2 - 558	Stope, Station 148 + 102' west	64
36	W43 - 2 - 558	Stope, Station 148 + 122' west	63
37	W43 - 2 - 558	Stope, Station 135 + 12' west	70
38	134	Station 133 + 4' west	62
39	131	Station 132 + 11' west	69
40	131	Station 131 + 26' east	76
41	130	Station 130 + 32' east,raise bottom	51
42	139	Station 138 + 2' west	69
43	139	Station 139 + 3' east	69
44	W43 - 2S - 551	Stope, East Face + 10'	54
45	W43 - 2S - 551	Stope, East Face + 26'	66
46	W43 - 2S - 551	Stope, East Face + 44'	65
47	W43 - 2S - 551	Stope, East Face + 77'	56
48	W43 - 2S - 551	Stope, East Face + 90'	62
49	W43 - 2S - 551	Stope, East Face + 113'	57
50	146	Station 154 + 57' west	62
51	154	Station 154A + 20' west	70
52	121	Face + 5' east	64
53	121	Face + 36' east	70
54	119	Face + 73' east	72
55	122	Station 122 + 10' to Station 120	68
56	107	Station 108 + 35' west	55
57	107	Station 108 + 67' west	43
58	107	Station 108 + 90' west	64
59	W43 - 2 - 558	Stope, Station 148 + 92' west	33
60	W43 - 2S - 551	Stope, East Face + 95' West	29
61	121	Face + 15' east	36
62	131	Station 131 + 30' east	42
63	139	Station 139 + 5' east	54

Note: Samples 59,60,61,62 & 63 are handpicked ore from bulk samples.

Total Webber	2,058
Total Huestis	2,090
TOTAL HUESTIS & WEBBER	4,148

HUESTIS: bulk samples for metallurgical testing

HUESTIS 4100 LEVEL

1 - Stope	H41 - 12 - 588, between chutes No 4 & 5	approx.	1 ton
2 - Ore shoot	H41 - 12 - 585, Station 713+10' west	approx.	2 tons
3 - Ore shoot	H41 - 12 - 585, Station 713+55' west	approx.	1,5 tons
4 - Ore shoot	H41 - 12 - 585, Station H97+38' west	approx.	1,5 tons

HUESTIS 4300 LEVEL

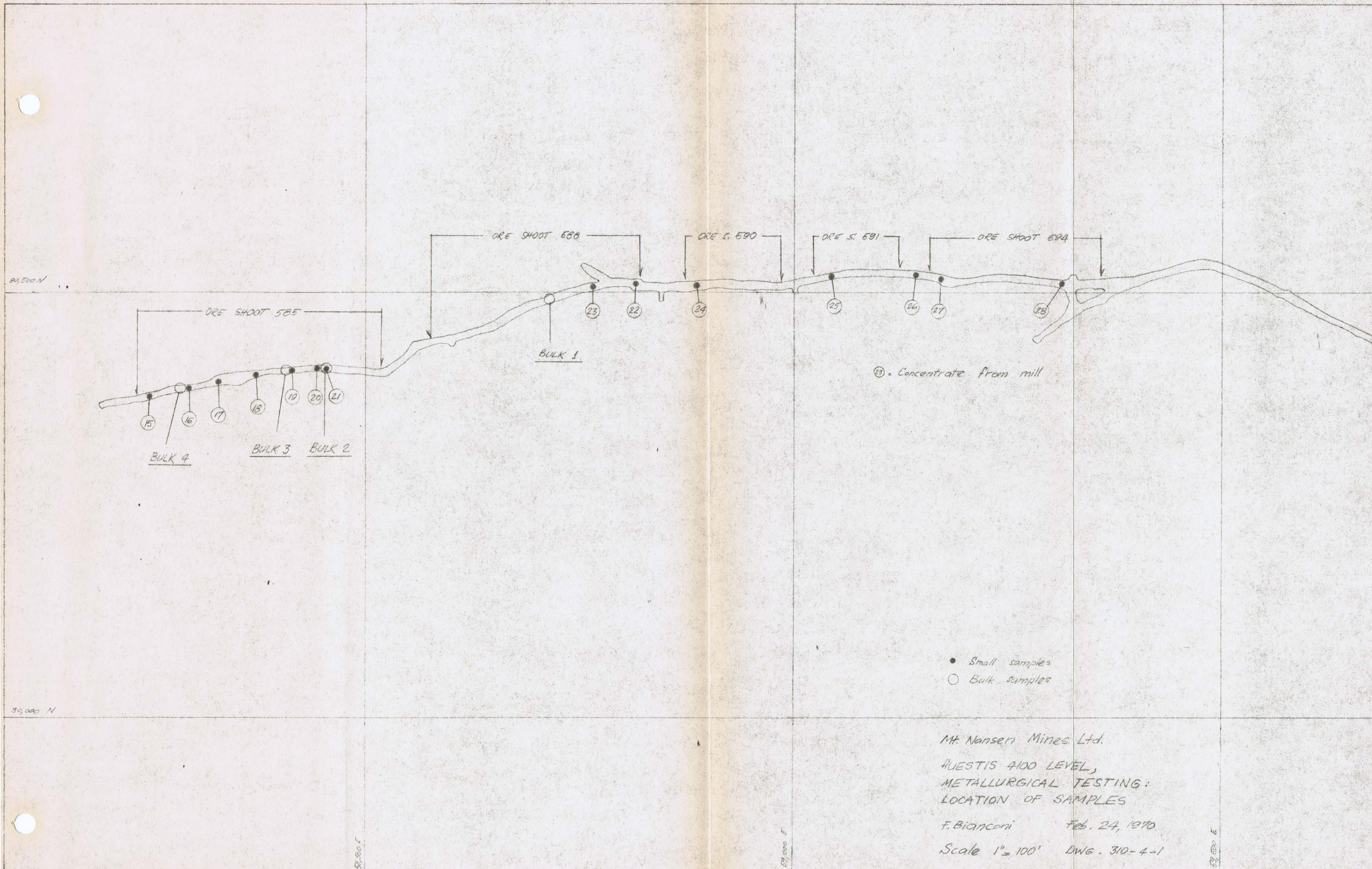
5 - Ore shoot	H43 - 12 - 588, Station H54+6' west	approx.	2 tons
6 - Ore shoot	H43 - 12 - 591, Station H20+23' west	approx.	2 tons
	Total	approx.	10 tons

WEBBER: bulk samples for metallurgical testing

WEBBER 4260 LEVEL

1 - Stope W43 - 2 - 558, Station 148+92' west	approx.	2 tons
2 - Stope W43 - 2S - 557, East Face + 95' west	approx.	0.5 tons
3 - Ore shoot 121, Face + 15' east	approx.	2.5 tons
4 - Ore shoot 131, Station 131 + 30' east	approx.	2.5 tons
5 - Ore shoot 139, Station 139 + 5' east	approx.	2.5 tons
Total	approx.	10.0 tons

TOTAL HUESTIS & WEBBER: approx. 20.0 tons

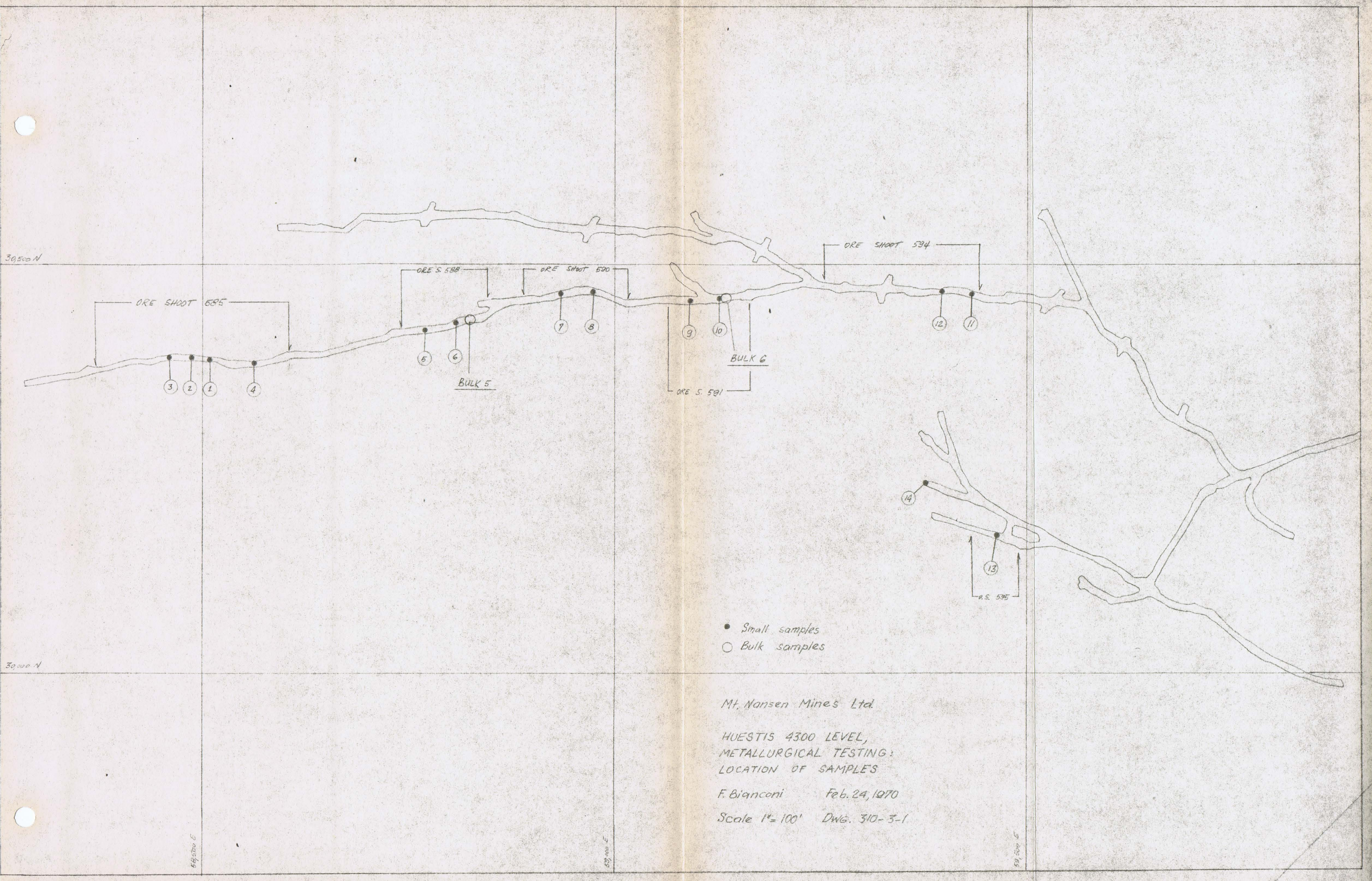


Mt. Nansen Mines Ltd.

HUESTIS 4100 LEVEL,
METALLURGICAL TESTING:
LOCATION OF SAMPLES

F. Bianconi Feb. 24, 1970

Scale 1" = 100' DWG. 310-4-1

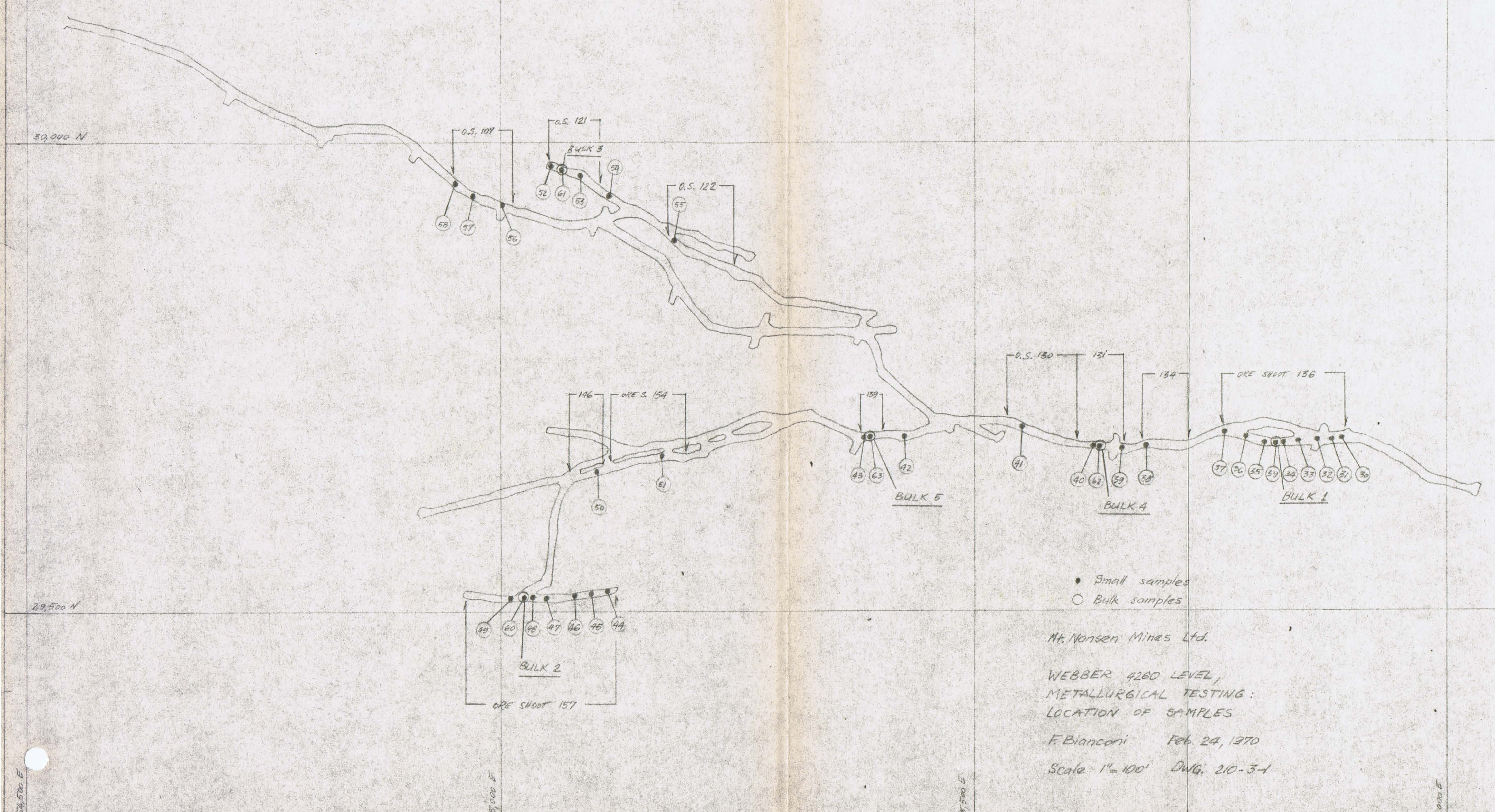


Mt. Nansen Mines Ltd.

HUESTIS 4300 LEVEL,
METALLURGICAL TESTING:
LOCATION OF SAMPLES

F. Bianconi Feb. 24, 1970

Scale 1" = 100' Dwg. 310-3-1



Mt. Nansen Mines Ltd.

WEBBER 4260 LEVEL,
METALLURGICAL TESTING:
LOCATION OF SAMPLES

F. Bianconi Feb. 24, 1970

Scale 1" = 100' DWG. 210-3-1

PHONE 684-7311

S&N Mine Management Consultants Ltd.

EXECUTIVE OFFICE 1300 MARINE BUILDING, 355 BURRARD STREET, VANCOUVER 1, B.C., CANADA

November 6th, 1969

12 XI 69

Mount Nansen Mines Limited
420 - 475 Howe Street
Vancouver 1, B.C.

Dear Sirs,

Re: Mount Nansen Mill

At your request, please find enclosed, my evaluation of the present condition of the Mount Nansen Mill and the repairs and changes required to make it operational and more efficient. As this portion of the report is based on memory there may be some minor omissions or discrepancies.

The condition of the cyanide building and machinery on site are noted together with a list of machinery required. Some of the missing equipment may be at Nelson Machinery. Some work done on tank layout and elevations are included with the report.

The conditions for Metallurgical Test Work are outlined. The extractive scheme is the result of observations made during plant operation and modified later by idea exchanges.

Yours very truly,

S & N MINE MANAGEMENT CONSULTANTS LTD.

C. W. Coffey

C. W. Coffey.

CWC/tp
Enc.

MOUNT NANSEN MILL

Present Condition and Changes Required

Mine Ore Transfer

Mine ore is received, through a grizzly, by a small hopper. The ore is fed from the hopper to #1 Conveyor by a modified 30" x 7' S.A. style B Pan feeder. This feeder is useless and should be replaced by a similar used feeder in good condition. No. 1 Conveyor is a 30" wide belt conveyor, 309' between centers with a lift of 73' 6". The belting on this conveyor is too light for the strains imposed on it and should be replaced with a suitable belt.

The pump-circulated hot water heating system to the Mine Ore Transfer House and No. 1 Conveyor gallery was a complete failure. This system should be replaced by a steam system, the steam being drawn from the nearest main. The condensate line should be placed beside the steam line in exposed areas and both insulated. Radiators should be lowered to prevent the freezing of spillage. No. 1 Conveyor Gallery would be heated by the escape of warm air from this area.

Crusher

No. 1 Conveyor discharges into the Coarse Ore Bin. The beam across the bottom opening at the Coarse Ore Bin should be removed, as the feed chute has been modified to relieve the pressure on the Syntron Feeder.

The Syntron Feeder is in good working condition, but the controls for this unit and the stop-start for No. 2 Conveyor should be moved to the operating platform.

No. 2 Conveyor, a 30" wide belt conveyor, 18' 4" between centers, with a lift of 5' 10", requires a new heavy duty belt. The cast iron head-pulley on this unit is breaking up and should be replaced. The replacement pulley is at the mine site.

The Bar Grizzly between No. 2 Conveyor and the Jaw Crusher should be replaced with a chute.

The 18" x 32" Telsmith Jaw Crusher is in good condition.

The discharge from the Jaw Crusher falls onto No. 3 Conveyor. No. 3 Conveyor is a 24" wide belt conveyor, 87' between centers, with a lift of 16' 3". This conveyor is in working condition.

No. 3 Conveyor discharges onto No. 4 Conveyor at the Transfer House. No. 4 Conveyor is a 24" wide belt conveyor, 56' between centers with a lift of 14' 3". This conveyor is in working condition.

A magnet should be purchased to protect the cone crusher from tramp iron. It should be hung above No. 3 or No. 4 Conveyors.

No. 4 Conveyor discharges onto a double deck 3' x 8' Dillon Screen. This screen is in working condition, but cracks are developing at the openings in the frame. The spare screen in the crusher house should be rebuilt.

The screen oversize discharges into a 3' Symons Standard Crusher. The crushed material falls onto No. 3 Conveyor. This crusher is in good operating condition.

The screen fines discharge onto No. 5 Conveyor. No. 5 Conveyor is a 18" wide belt conveyor, and is 134' between centers, with a lift of 31'. This conveyor is in working condition.

No. 5 Conveyor discharges onto No. 6 Conveyor, located at the top of the Fine Ore Bins. No. 6 Conveyor is 18" wide horizontal belt conveyor, 14' between centers. This conveyor is in good working condition. There is at the mine site replacement belting for the 18" wide and 24" wide conveyors.

The machinery in the Crushing Plant is interlocked electrically in the usual manner except for the cone crusher. The switch gear on the cone crusher gave a great deal of trouble and should be repaired or replaced. No. 6 Conveyor must be started from the top of the bins. This switch has an interlock by-pass, which must be depressed when changing direction at No. 6 Conveyor.

A Dust Control System should be installed in the Crushing Plant.

The heating system in the Crushing Plant was also a failure due to long steam lines, a condensate return system, and improper placement of some radiators. The steam should be drawn from the main, near or in the warehouse and run directly to the Crushing Plant and then to the Transfer House at slight upward angle. Condensate lines should be placed close to the steam lines and insulated by the same cover in exposed areas. The large radiators in the Crushing Plant should be moved to the lower floor and some of the smaller radiators moved to the upper deck. The two radiators at the Coarse Ore Bin are correctly placed.

Ramsey Weightometers should be installed on these conveyors.

The chutes from the feed conveyors to the ball mills should be rebuilt and rubber lined to increase their life.

Both mills should be equipped with rubber lined drum feeders.

No. 1 Ball, an 8' x 48" Hardinge, is equipped with steel liners. New liners for this mill are at the mine site. The liners should be checked to see if this set is complete. There are no liner bolts. The synchronous motor on this mill has ring lubricated babbitt bearing. These bearings are prone to heat unless fed an excessive amount of oil. They should be re-babbitted with nickle babbitt, care being taken to provide oil return slots on the lower half of the bearing.

No. 2 Ball Mill, a 7' x 60" Hardinge, is equipped with new rubber liners.

The discharge boxes of both mills require repairs.

The 4 x 3 5RL-C Ball Mill Pumps are in working condition. A spare shaft should be obtained for these pumps (Denver).

The apex liner of No. 2 Ball Mill cyclone should be replaced.

No. 1 Conditioner requires a new three bladed ships-type impeller (Denver). Some patching is required on the metal tank.

No. 2 Conditioner is in working condition.

The Denver #18 Sp. Flotation Machines are in working condition, except for one shaft assembly that requires new bearings.

The thickners are in working condition.

The 1 3/4 x 1 1/2 SRL Concentrate Pump is in working condition. A new impeller should be ordered (Denver).

The Denver Filter requires a new drive gear.

The vacuum pumps presently on site are worn out and so old that parts are expensive and hard to obtain. The purchase of new pump is indicated, but its size should be governed by the projected requirements of the mill.

The Blower is in good working condition.

The Filtrate Pump is in good working order, but requires a new coupling.

The 2 " x 2" SRL Ball Mill Sump Pump (A.C.) is in good working condition.

The 3" x 3" SRL Tailing Pump (A.C.) is in good condition.

The 2" x 2" SRL-V Pump is old, expensive to maintain, and presently requires a new shaft. A newer SRL-V should be purchased.

On site, and partially installed is a 55 Marcy Mill for regrinding rougher concentrates. Britton Research Ltd. indicated that as the concentrate showed considerable mineral gangue attachment an upgrading concentrate could be achieved by grinding the rougher concentrate to minus 200 mesh followed by cleaning. As all components are at the mine site for this addition, some consideration should be given to completing this addition.

If the box method of shipping concentrate is retained, the roller system from the scale should be extended so that the concentrate boxes can be located under the mono-rail. The mono-rail should be motorized. But due to the high cost of shipping by this method, alternative systems should be investigated.

The Tailing Dam requires repair. Only clay materials should be used.

Cyanide Plant

The building (a prefab 96' x 50' metal structure) has been erected. The retaining wall and thickner tunnel has been poured, for thickner bench. No other work has been done on this plant. If test indicates the feasibility of cyaniding Mt. Nansen ores or flotation tails certain preliminaries will be required. An access door should be cut in the back of the mill, and a road should cut to this point. This would permit the entry of fill and concrete required for the completion of the thickner bench, and would later serve as the entry point for lime and cyanide necessary for processing. This area appears to be on bedrock and should present no support problems.

The next level designated the Agitator area is partially on bedrock and partially on muck. The muck should be removed and after the area is filled, capped with a thick layer of re-enforced concrete. A re-enforced retaining wall, four and one half feet high above the floor, at the lower end of the agitator area would form a sump, large enough to contain the contents of anyone of the tanks containing pregnant solutions.

The lowest level is on muck with underlying permafrost. This area should also be deepened and filled before capping. Concrete under the barren solution tank and filter should be thickened and re-enforced.

Due to equipment changes, the pulp level in the thickner appears to be 2' 10" above the pulp level in No. 3 Agitator, when the agitators are raised as high as possible. A pump at the discharge of No. 3 Agitator appears to be a cheap alternative to lowering the existing concrete work.

Equipment at the Mine Site

Thickner - tank 30'ø x 10' new, complete with launder.
Bridging and Mechanism - Wemco S/N 5319756, appears to be in good condition.

No. 1 Agitator - tank 22' ø x 18', not all at mine site, bottom staves rotten, new tank may be cheaper than trying to patch this one.
Bridging and Mechanism - Dorr Center Lift S/N DOL 215-2, appears to be complete and in good condition.

No. 2 and No. 3 Agitators - tank 16' ø x 16', some signs of rot on the bottom, some patching appears to have been done. May pay to order some nominal 3" x 12" planks.
Bridging and Mechanisms - Denver S/N 7480-1. .No motors, otherwise appear complete and in good condition, aeriation achieved by 2 stationary air lifts per tank, rakes appear to be of local construction and require some repairs.

Pregnant Solution Tank - 16'ø x 16' same condition as above.

Barren Solution Tank - 16'ø x 16' same condition as above.

Dorco Pressure Pump - 3", S/N US2838 complete with motor.

C.I.R. Vacuum Pump - Size 4" x 4", P63 BAC #28179.

Zinc Dust Feeder - 6' x 8" with cone.

Required:

1 only Clarifier Tank with leaves
1 only De-aeriating Tank
1 only Filtrate Pump for above
1 only Zinc Emulsion Pump
1 only Preg. Solution Pump
1 only Barren Solution Pump
Precipitation Units or Press
1 only Filter complete with winding gear
1 only Repulper
1 only Vacuum Tank
1 only Dewatering Tank
1 only Vacuum Pump (see previous notes)
1 only Compressor or use Mine Air
1 only Drying rack (can be built at Mine Site)
1 only Rootes Blower or pressure reduction Valve

Metallurgy

It would appear likely that the Huestis ores and the Webber ores at depth, will be similar. At present the Webber ores are highly oxidized, while Huestis ores are mainly sulphides. Any successful extraction scheme must be able to deal with both types of ore, and to produce high grade marketable products. To this end, additional metallurgical testing is required.

Samples to be tested must be representative of all ores to be treated in the concentrator. Results of such tests should be evaluated economically and not metallurgically. Tests should be made within the limiting factors of the existing and extendable mill. These tests should ascertain the feasibility of definite extractive schemes.

As argentiferous sulphide ores generally float best at their natural PH, the production of a high grade silver concentrate containing little arsenic may be possible. The second stage would involve the flotation of pyrite and arsenopyrite. This latter concentrate could be roasted and leached or shipped depending on grade.

It is not likely that the existing supply of Webber ore would respond economically to the above treatment, even with an extended flotation circuit and sulphiding of the oxides. The tails from sulphide flotation would possibly respond to cyanidation.

If the above observations are valid, an extractive scheme flexible enough to work on both ores could be as follows:

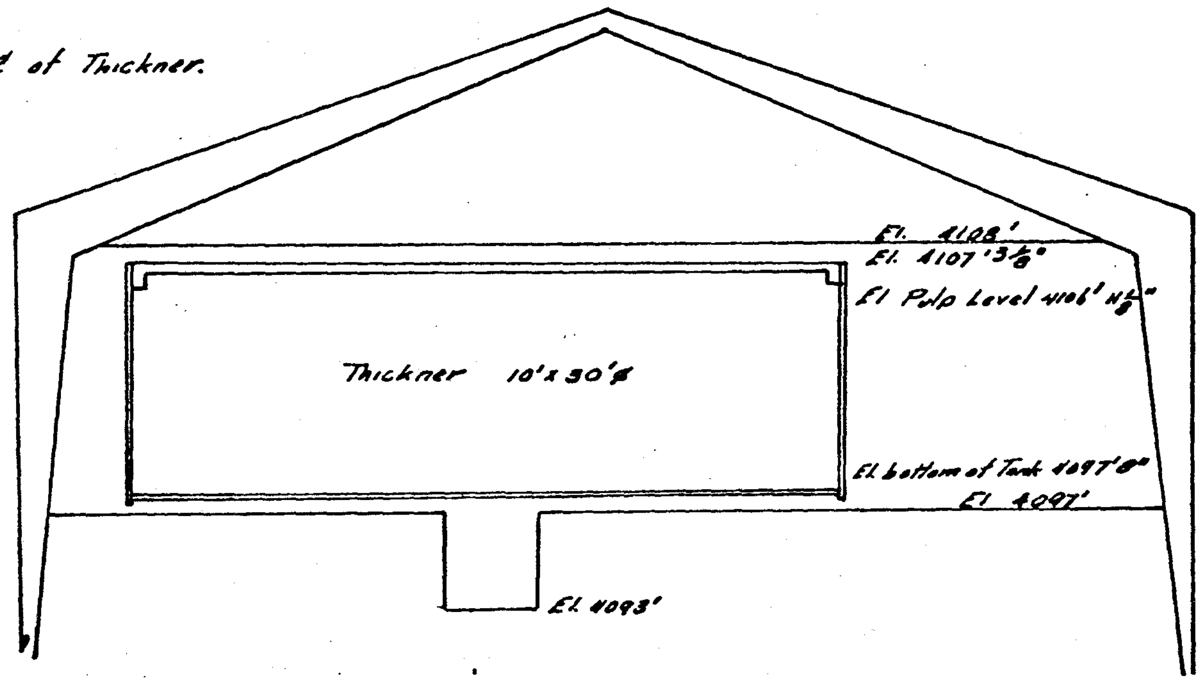
(1) Extraction of a silver bearing rougher concentrate.
(2) Regrinding this concentrate to minus 200 mesh and up-grading its silver content by cleaning and recleaning, aided by depression of iron minerals in the cleaner circuit. The resulting silver tails could be in the order of ± 3 oz. Ag per ton, depending on the heads.
(3) Removal of the gold bearing cyanicides by flotation, while depressing the silver minerals.
(4) Roasting the pyrite and arsenopyrite concentrate.
(5) Cyanidation of flotation tails and roaster products.

In the event that this, or some variation of this scheme, proves practical, mixing of sulphide and oxide ore would be possible, even desirable in order to limit the amount of silver reporting for cyanidation. In practice, the relationship of concentrate grade to the recovery in the cyanide plant would be the key to optimum economic recovery.

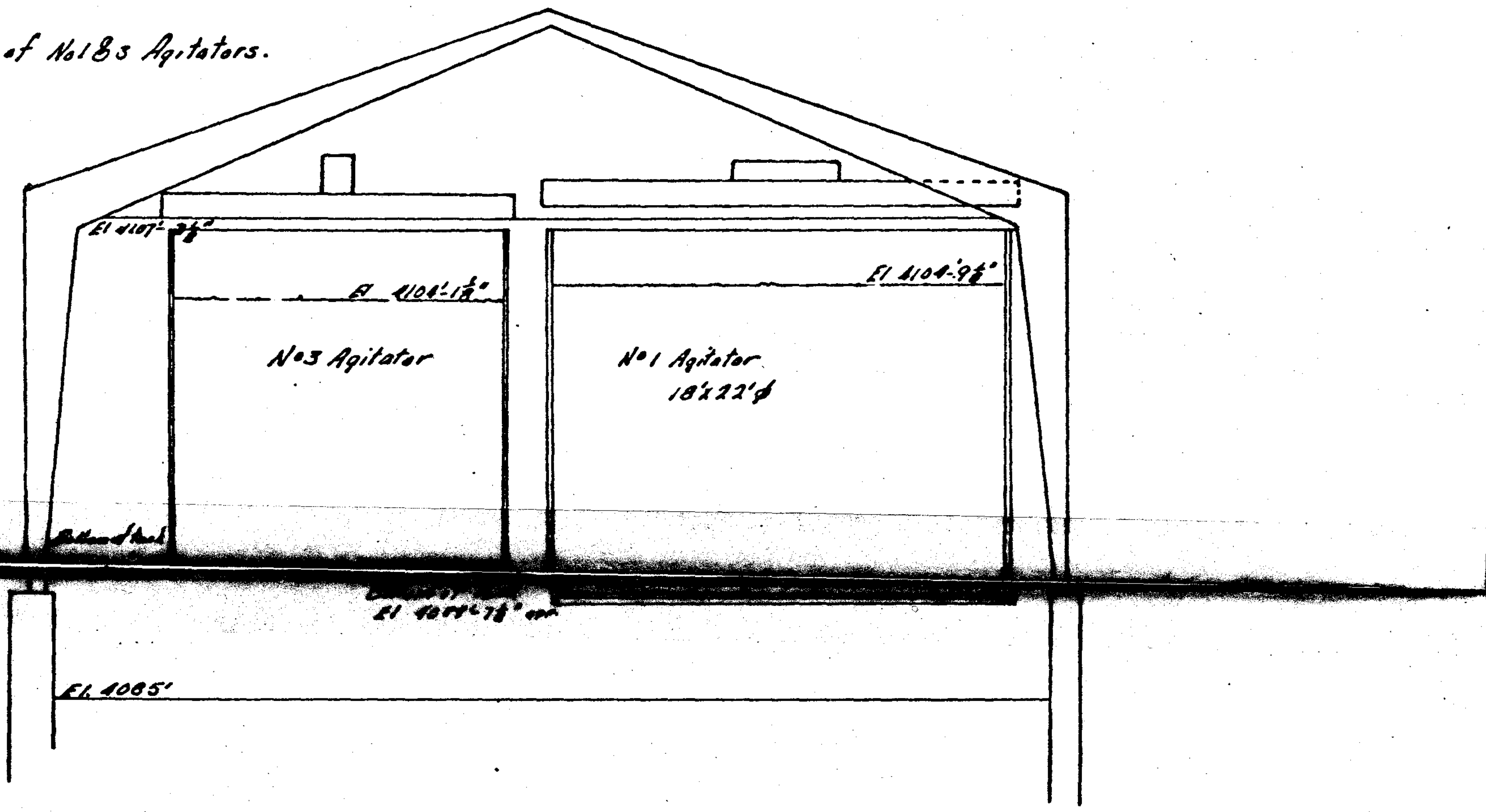
If this scheme or any scheme involving cyanidation proves practical, at least one test should be run using mill water. This could preclude unpleasant surprises.

MOUN_ NANSEN MINES (preliminary Cyanide plant. showing pulp level with agitators raised as high possible)

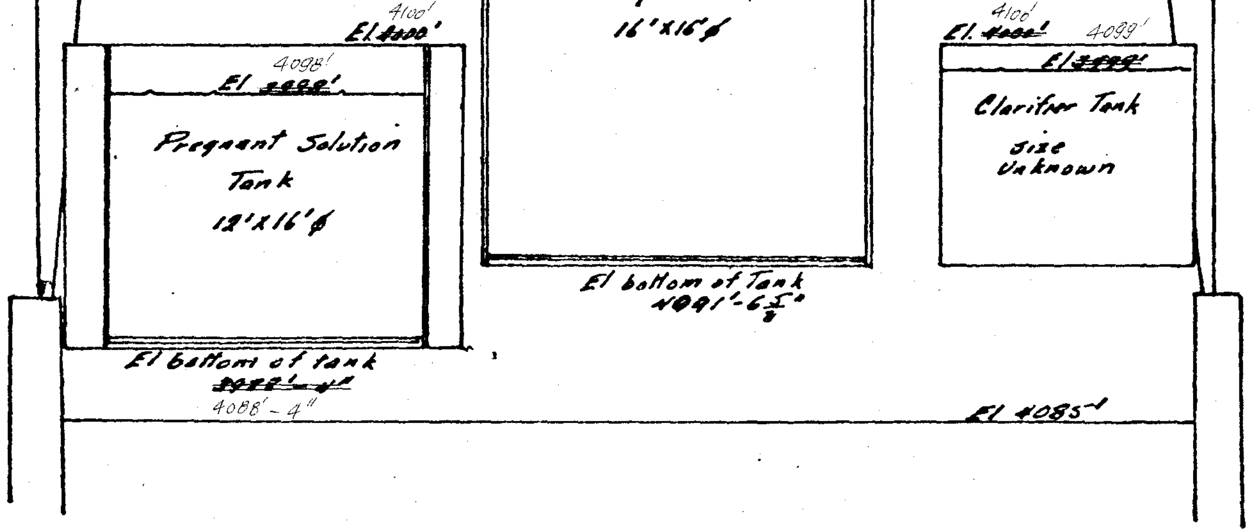
Section through ℄ of Thickner.



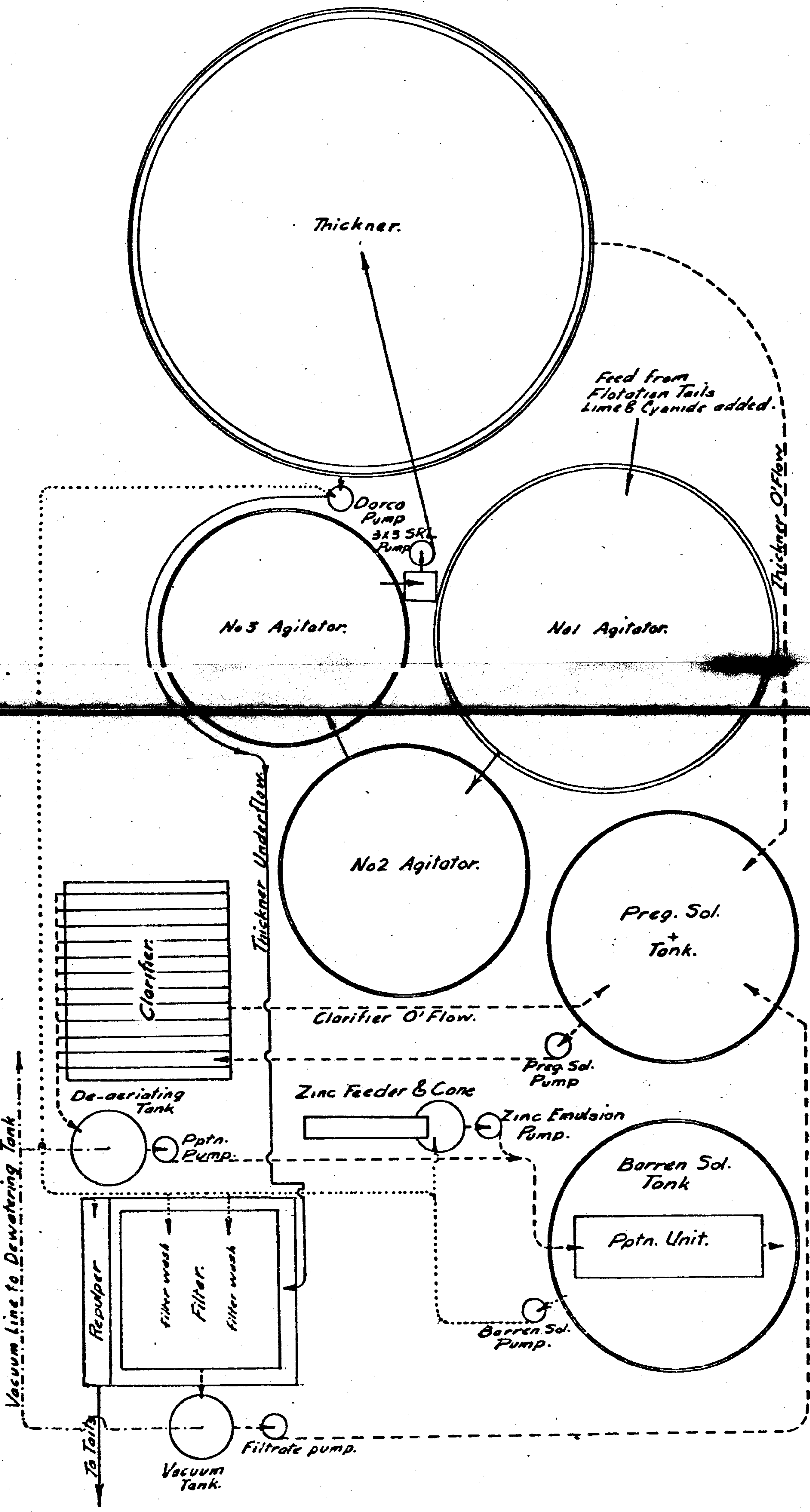
Section through ℄s of No1 & 3 Agitators.



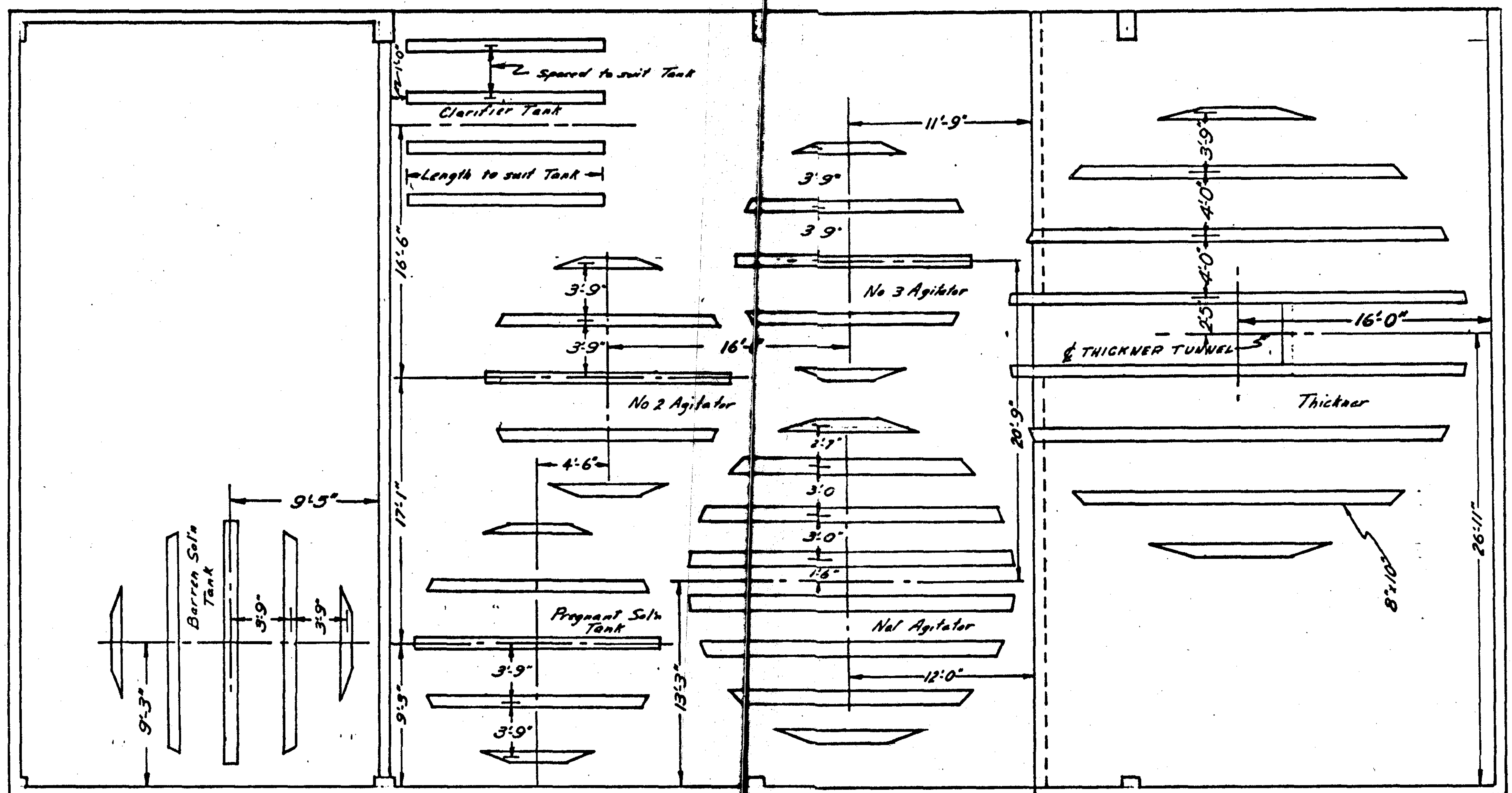
Section through ℄ No 2 Agitator, Clarifier & Preg. Sol. Tank.

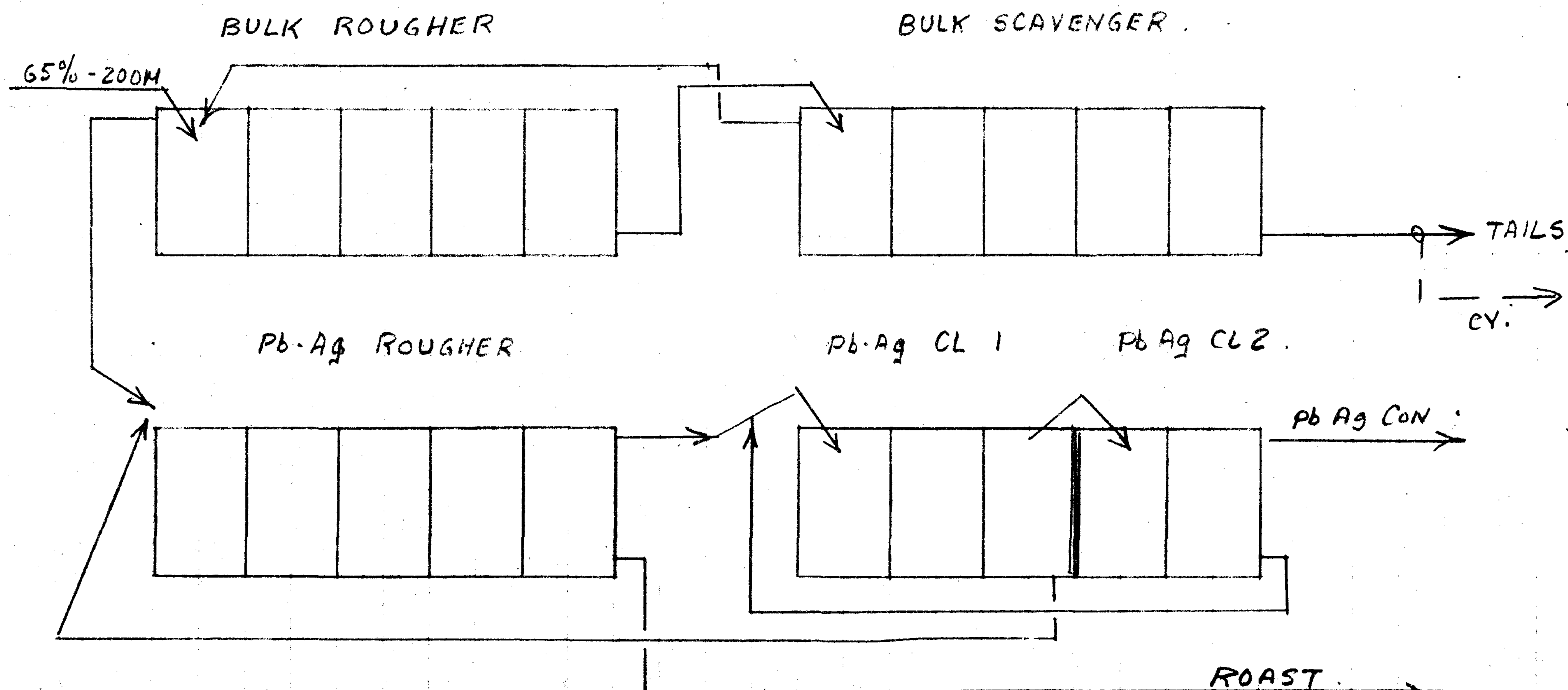


MOUNT NANSEN MINES (preliminary flowsheet for Cyanide Plant)



BY.............................DATE....................... SUBJECT Mt. Hansen Cyanide Plant SHEET NO.............OF
CHKD. BY...............DATE....................... Location of Tank and Equipment JOB NO.
Bents




BULK ROUGHER
BULK SCAVENGER.
65% -200M
TAILS
CY.
Pb·Ag ROUGHER
Pb·Ag CL 1
Pb Ag CL 2.
Pb Ag CoN
ROAST