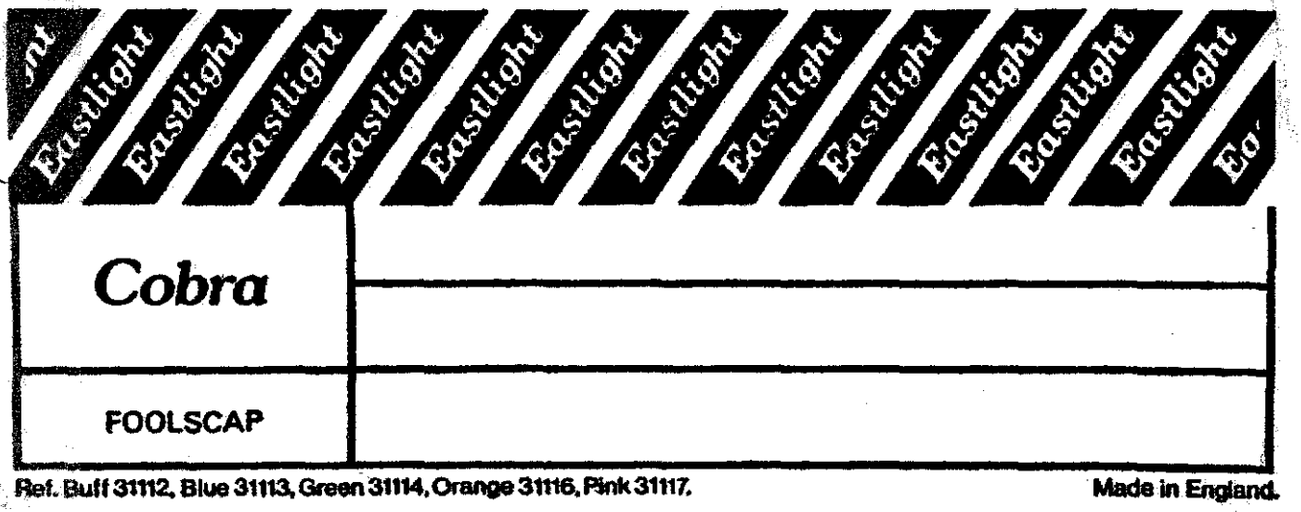
Amenability + Mineralogical Reports.
January 10, 1972 to August 3, 1973
Kara Tungsten Prospect.
EL 17/68
Tas.

662001

Union Carbide Corporation / Research.
+ Development Department - Mining.
and Metals Division
for
Australia and New Zealand Exploration Company

73-972



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INTERNAL CORRESPONDENCE

UNION CARBIDE EXPLORATION CORPORATION • 270 PARK AVENUE, NEW YORK, NEW YORK 10017

To (Name) D. L. Mathias, Jr.
Division Exploration
Location Sydney, Australia

Date August 3, 1973

Originating Dept. Exploration - New York

Answering letter date

Copy to T S Ary
A. W. Heuck

Subject TASMINEX

Dear Dave:

Enclosed are copies of all the reports issued by our R & D group in Niagara Falls concerning the Tasminex project.

In addition, there are copies of all the U.C.C. invoices for costs of the R & D group with details supporting those charges.

If we can be of any other service in assisting you to resolve this problem, please let us know.

Regards,

L. A. Wright

L. A. Wright

662003

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. A. W. Heuck (2)
Division UCEX
Location Union Carbide Corporation
270 Park Ave.
New York, NY 10017

Date January 30, 1973

Originating Dept. Research and Development Department

Answering letter date

Copy to Messrs. R. J. Klotzbach
M. Stern
L. A. Wright

Subject Examination of Tasminex Core Samples Submitted by A.W. Heuck

Dear Art:

While awaiting the drill cores from the proposed 1973 Tasminex program, we gave a cursory examination and evaluation of the samples you returned with last November. These samples consisted of portions from drill holes D-113, D-117, D-121, D-125, and D-127. The description of these holes is supplied in the attached memorandum by E. E. Anderson.

Our objective was to compare the magnetic processing amenability on the basis of (1) oxidation state, and (2) ore grade. Unfortunately, none of the holes contained ore with any degree of oxidation so all samples were classified as unoxidized. Because the WO_3 grade varied significantly, all samples were composited on this basis. The exception, of course, is drill hole D-117 in which the ore occurred in a matrix other than magnetite.

The results of the magnetic amenability tests are reported in the attached memorandum by N. L. Grauerholz. These results show greater than 90% WO_3 recovery in the non-magnetic fraction with the magnetite fraction containing less than 0.10% WO_3, indicating that the unoxidized ore is truly amenable to magnetic beneficiation. Based on this information, one could calculate that the unoxidized ore could be processed as follows:

	% Wt.	% WO_3	Units	% WO_3 Recovery
Non-magnetic	25	2.10	.525	93
Magnetic	75	0.05	.0375	7
Ore	100	0.56	0.56	100

This is considerably better than obtained in our test work performed on Sample "C" (unoxidized ore) reported in September of 1972. The difference in amenability is attributed to the content of the brown oxidized ore in Sample "C". Table I of this letter contains a material balance comparison of Sample "C" as compared to samples from these tests. In this material balance I have assumed a 500-ton a day mill processing 50% oxidized ore (no upgrading) and 50% unoxidized ore. The significant difference is in the overall recovery of 87.4% for these samples compared to a calculated 80.3% for Sample "C".

The second half of Table I shows the relative significance of the improved amenability in terms of cost per unit WO_3 recovered. Because this is an order of magnitude, it should be pointed out that the cost difference is only relative.

We are still awaiting the samples from the 1973 drilling program and upon receipt of these samples we will evaluate them in a similar manner. Hopefully, these samples will contain the oxidized, unoxidized, and transitional ore so that a better perspective can be gained.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

TABLE I

TASMINEX PROJECT

50% of Ore Upgraded by Magnetic Separation

Material Balance

	Sample "C"				Sample D-121-125			
	Tons	Units	% Rec.	WO_3 Grade	Tons	Units	% Rec.	WO_3 Grade
Grind	500	330		0.76/ 0.56	500	330		0.76/ 0.56
Mag. Sep.	250/ 68	140/ 104	74	1.65	250/ 625	140/ 130	93	2.10
Roast	318	294/ 270	91.5	0.93	312.5	320/ 294	92	1.02
Leach	-	270/ 265	98	-	-	294/ 288.5	98	
Product	-	265/ 330	80.3	70		288.5/ 330	87.4	70

Order of Magnitude Cost Estimate

	$ Per Unit WO_3 Recovered as Syn. Scheelite	
	Sample "C"	Sample D-121-125
Mining Costs	6.79	6.24
Mill Labor Costs	4.25	3.91
Salt Roasting	9.22	8.32
Other Mill Costs	14.29	13.13
Total Operating Costs	34.55	31.60
Depreciation (Assume $6MM Tot. Cap.)	6.86	6.30
Total Cost Per Unit Recovered	41.41	37.90

1/29/73

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RECEIVED
JAN 29 1973
R G WOOLERY

Research and Development Department
Mining and Metals Division - UCC
Niagara Falls, New York
January 29, 1973

M E M O R A N D U M

TO: Messrs. N. L. Grauerholz
D. J. Hansen
R. G. Woolery

COPY: R. & D. File

FROM: E. E. Anderson

SUBJECT: Preparation of Composites from "Scheelite Only" Zone of Tasminex, Kara #1 Drill Hole Samples

Cuttings from portions of drill holes D113, D117, D121, D125, and D127 were received at the Niagara Falls Laboratory November 17, 1972 for beneficiation tests. Mineralogic characterizations of these ores were begun with the intention of using them as criteria to define different composites for beneficiation tests.

After a preliminary visual examination of the samples, four basic criteria were developed for the purpose of compositing the samples. First, it was decided to keep each hole separate rather than combining similarly appearing samples from a number of holes. It was felt that the data could always be combined for calculated "super composites." The second criterion was color or color changes between samples within a hole. Color changes usually reflect changes in mineralogy. We were particularly looking for changes in oxidation and hydration of the iron oxide minerals because of the previously observed association of these changes with changes from scheelite to non-scheelite ore. The third criterion is similar to the second, but consists of changes determined by actual mineral identifications. The fourth criterion was sample grade as taken from the Australia and New Zealand Exploration Company's (ANZECO) drilling logs.

COMPOSITE #1

Drill Hole 113, 98.0-107.0 Feet

Color differences were deemed insignificant. Therefore, the three adjacent highest grade samples were chosen in order to make a composite grade as close as possible to the .61% WO_3 ore grade for the Kara #1 property as determined by ANZECO. The unweighted average analysis for this composite was .57% WO_3.

COMPOSITE #2

Drill Hole 117, 58.0-70.2 Feet

As in drill hole 113, the samples from drill hole 117 did not differ from each other significantly in color. Five adjacent samples of highest grade were combined as representing a mineable thickness averaging .93% WO_3. The

sample above this material assayed .01% W and the one below assayed 0.006% W. The bulk mineralogy of composite #2 does differ significantly from those of the other composites. It consists predominantly of the hydro-garnet mineral, hibschite, $Ca_3Al_2Si_2(OH)_4O_8$, with lesser amounts of diopside, $CaMgSi_2O_6$, red fluorescent calcite, $CaCO_3$, magnetite, Fe_3O_4, and scheelite, $CaWO_4$.

This and other occurrences of significant amounts of tungsten with relatively small amounts of magnetite are rare in the Kara samples examined to date. However, they indicate that the relationship magnetite:tungsten is not universal in the Kara district and that the absence of a magnetic anomaly does not in itself indicate lack of tungsten.

COMPOSITE #3

Drill Hole 125, 160.7-190.2 Feet

Again the samples from drill hole 125 did not appear to vary significantly, so again tungsten assay was the compositing criterion employed. The composite consisted of 10 adjacent samples ranging in grade from .12% to 1.86% W for an unweighted average of .55% WO_3. Assays for the sample above and below the composited interval were subeconomic .06% W.

The samples from hole 121 did show some variations in color which we thought might reflect significant changes in mineralogy. X-ray diffraction patterns were made for each sample and the relative amounts of the major constituents determined. While variations in the proportions of the constituent minerals did correspond to color changes of the bulk samples, they were deemed insignificant in terms of predicted beneficiation response. Since this was the most extensively sampled hole received, it was broken into three composites on the basis of assay. The upper two composites (4 and 5) are relatively high grade, whereas the deeper composite (6) is lower grade. The tungsten grade of the material below composite 6 would evidently preclude economic recovery.

COMPOSITE #4

Drill Hole 121, 113.0-141.0 Feet

Unweighted average grade = 1.11% WO_3.

COMPOSITE #5

Drill Hole 121, 141.0-171.3 Feet

Unweighted average grade = 1.02% WO_3.

COMPOSITE #6

Drill Hole 121, 171.3-187.0 Feet

Unweighted average grade = .43% WO_3.

Drill hole 127 was not composited because of its low grade. We have 9 samples from 220.5 to 240.5 feet which average .30% WO_3. These could be composited if it is desirable to obtain beneficiation data on material of this grade.

E. E. Anderson

EEA:ms

Copied for KPK 1/07/10 use

RECEIVED
JAN 9 1973
R G WOOLERY

662009

Research and Development Department
Niagara Falls, New York
January 5, 1973

M E M O R A N D U M

TO: Messrs. E. E. Anderson
D. J. Hansen
R. G. Woolery

COPY: R & D. File

FROM: N. L. Grauerholz

SUBJECT: Magnetic Amenability of Deeper Tasminex Ore Samples
(Tungsten Ore/Raw Material Processing)

SUMMARY

The drill core samples obtained from deeper unoxidized ore zones in the Tasminex tungsten property were beneficiated by magnetic methods. The amount of magnetic material in these samples varied from 6.9% to 76.1% of the total weight in these samples. The magnetic material contained a very small fraction of the total tungsten and indicated that this type of material is very amenable to upgrading by magnetic beneficiation.

Drill Hole D-117, 24 through 28, contained substantially less magnetic material than the other samples. This is significant from both a geological and mineral beneficiation viewpoint. Other beneficiation methods should be investigated on this sample in more detail.

DISCUSSION

A set of drill hole core samples was received at the Research and Development facilities on November 17, 1972. These samples from drill holes from the Kara No. 1 section of the Tasminex ore body represented deeper sections of the ore body. These samples were to be examined mineralogically to determine variations from previous samples.

A series of composite samples representing the higher tungsten content ore zones from each drill hole were assembled. These sample composites were magnetically upgraded to determine their amenability to magnetic separation.

This set of drill core samples was assigned our number, OR-600, for identification purposes. Each composite made by the R & D mineralogical laboratory is further identified as follows:

Memo re Magnetic Amenability of Deeper Tasminex Ore Samples - 2 - January 5, 1973

Sample No.	UCEX Designation	R & D Designation
1	D-113 - 24, 25, 26	Comp. 1903-32-1
2	D-117 - 24 through 28	Comp. 1903-32-2
3	D-125 - 45 through 54	Comp. 1903-32-3
4	D-121 - 39 through 48	Comp. 1903-33-1
5	D-121 - 49 through 58	Comp. 1903-33-2
6	D-121 - 59 through 63	Comp. 1903-33-3

Each sample composite was comprised of equal splits from each footage increment as received. The sample splits were combined and crushed to minus 8 mesh. The minus 8-mesh composite was then riffle split to give a 200-400 gram portion which was further crushed to 100 mesh for an analytical sample and a test charge for Davis tube tests. The remaining fraction of minus 8-mesh material has been retained for further beneficiation tests if desired.

Two 25-gram charges of the minus 100-mesh composite of each sample were tested with the Davis tube. The magnetic and nonmagnetic fractions from each test charge were combined for analysis. The large amount of magnetite in the ore limited each test charge for the Davis tube to approximately 25 grams. Wash water was 300 cc./min. The amperage setting was 2.0 amps. and the speed was set at approximately 100 oscillations per minute. The test period was not timed but was conducted until the wash water was clear.

The results of each test are shown on the attached data page. The upgrading accomplished by magnetic separation is substantial with all the samples treated except for Sample No. 2. This particle sample is much lower in magnetite than other test samples submitted from this deposit. The tungsten content is high and may be of mineralogical significance in extending the ore zones in this deposit.

The tungsten content in the magnetic fractions of these drill core samples is low. This has in the past indicated that the nonmagnetic fraction is amenable to further upgrading by flotation with a high degree of recovery. Additional beneficiation tests on these samples is recommended. Sample No. 2 is especially favored for additional beneficiation tests because it is low in magnetite but high in tungsten and may be indicative of a different beneficiation problem.

N. L. Grauerholz

N. L. Grauerholz

NLG:et
Attach.

Magnetic Amenability of Deeper Tasminex Ore Samples

Davis Tube Tests

Sample	Product	% Weight	Assay, % WO_3		Dist., % WO_3
1	Magnetic	50.0	0.09		9.9
	Nonmagnetic	50.0	0.82		90.1
	Calc. Head	100.0	0.46	1.8	100.0
2	Magnetic	6.9	0.10		0.5
	Nonmagnetic	93.1	1.58		99.5
	Calc. Head	100.0	1.48		100.0
3	Magnetic	60.6	0.02		1.9
	Nonmagnetic	39.4	1.62		98.1
	Calc. Head	100.0	0.65	0.5	100.0
4	Magnetic	70.7	0.08		5.4
	Nonmagnetic	29.3	3.40		94.6
	Calc. Head	100.0	1.05	3.2?	100.0
5	Magnetic	76.1	0.06		5.5
	Nonmagnetic	23.9	3.26		94.5
	Calc. Head	100.0	0.82	4.0	100.0
6	Magnetic	74.8	0.05		8.6
	Nonmagnetic	25.2	1.58		91.4
	Calc. Head	100.0	0.44	3.6	100.0

662012

Research and Development Department
P. O. Box 579, Niagara Falls, N. Y.
October 5, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: Messrs. E. E. Anderson
D. J. Hansen
R & D File

FROM: N. L. Grauerholz

SUJBECT: Recovery of Tungsten Values from Tasminex Ore Samples
(Tungsten Ore/Raw Material Processing)

This memorandum is written, not in the perspective of presenting an economical recovery method for recovering tungsten from the Tasminex oxidized ore, but to present alternative methods that may be applicable to this ore. The ore body is of too much interest to consider dropping the project because conventional treatment methods have not been successful or not economical.

Nature has oxidized the tungsten content of the ore body to form a vagrant tungsten ion and/or tungstic acid. Literature reports show that the difficult part of recovering tungsten products from ores is the "opening up" of the ores or releasing the WO_3 radical from the minerals where it occurs either as an iron or calcium tungstate. The WO_3 radical has already been released in the oxidized Tasminex ore and should be available for straight leaching. The chemistry of tungsten solubilization is not easy, not specific, not straightforward, but possible.

Many leaching tests were conducted on a "bootleg" basis on the Tasminex ore in conjunction with the magnetic and flotation tests conducted on the various ore samples submitted. These tests were all unsuccessful except for one that used an entirely different technique. The other tests all utilized chemicals on the basic side where the alkaline tungstates are soluble. One test was conducted on the acid side using sodium dithionate and hydrochloric acid.

The lead for this approach was found in Mellor[1] where considerable discussion on reaction of tungsten trioxide with various chemicals is found. He states that "O. Brunck found that in acid solution the trioxide is reduced by sodium hyposulphite."

(1) A Comprehensive Treatise on Inorganic and Theoretical Chemistry by W. Mellor.

Fiegl[2] states that peracid formers such as tungstates form the pertungstates and leads to the formation of sulfates rather than the tetrathionate that is the normal reaction with acids and sodium thiosulfate.

A 50-gram sample of Tasminex ore, Category A, was heated for 2 hours at 150°F with 0.10 lb./ton HCl and 2.5 grams of sodium dithionate. The pulp pH was raised to 10.0 (CaOH) was filtered. The conditions of this test with minus 8 mesh ore recovered 65% of the tungsten. The test has been repeated on a minus 100 mesh fraction of the ore to determine if additional ore can be recovered.

This memorandum is written solely to present a possible solution to recovery of the oxidized Tasminex ore. The solubilization of the non-scheelite tungsten in Tasminex is possible without roasting or pressure digestion and testwork along these lines should be attempted before abandoning the project.

(2) Specific, Selective, Sensitive Reactions by Fritz Fiegl.

Na Hyposulfite

N. L. Grauerholz

NLG:dg

$Na_2S_2O_4 \cdot 2H_2O$ - @ ~ 8¢/lb

$\frac{2.5\,gm}{50\,gm} = \frac{100\,gm}{2000\,gm} = 100\#/ton$ @ 8¢ = $8.00/ton

UNION CARBIDE

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date September 25, 1972

Originating Dept. *Research and Development Department*

Answering letter date

Copy to Messrs. A. W. Heuck (2)
J. C. Stephenson
M. Stern

Subject Tasminex Project
Order of Magnitude Cost Estimate

Dear Lew:

We have recently completed our laboratory evaluation of the Tasminex composite samples and as you are aware, the metallurgical response was quite different from the first set of samples received. Two factors most influenced in the recent tests were (1) amenability to upgrading and (2) the salt requirement per unit of WO_3. The higher salt requirement is largely due to the poor upgrading characteristics of the ore resulting in greater tonnage at lower WO_3 grade to the salt roast.

In light of these developments it seemed timely to prepare an order of magnitude cost estimate of the Tasminex project. There was not a great deal of effort given to this cost estimate as there seems to be some question as to the representative quality of the samples used. The capital costs were derived by estimating individual circuits in the process rather than from a detailed equipment list. (See Table II) Milling costs were estimated from current requirements at Arkansas and Rifle and the mining costs were supplied by L. A. Wright. (See Table III)

Three probable routes were evaluated and are referred to as (A) whole ore roast, (B) 50% of the ore (unoxidized portion) upgraded by magnetic separation, and (C) same as (B) except the non-magnetic concentrate is subjected to further upgrading by flotation. A proposed flowsheet incorporating all three alternatives is presented in Figure 1. A calculated material balance for all three routes (based on laboratory data) is shown in Table I.

This estimate is summarized on the attached summary sheet and shows the following estimated costs for the three routes proposed.

	Cost per Unit WO_3 Recovered
Alternate "A"	$44.65
Alternate "B"	41.41
Alternate "C"	42.63

This indicates it is most economical to upgrade the unoxidized ore prior to roasting (Alt. B). However, in this estimate 50% of the ore was assumed to be unoxidized and this criteria has yet to be determined. The cost data also show that the added flotation step is not economical.

These costs also indicate that a > $50.00 per unit WO_3 value F.O.B. Tasmania, would be required for this project to be viable.

A number of factors, however, could influence these costs and produce a more attractive prospect. These include.

1. Increased plant capacity
2. Increased average ore grade by selective mining
3. Selectively feed ore amenable to upgrading
4. Trim this cost estimate. (I suspect there is an equal chance that the cost would increase after a detailed study.)
5. Improve metallurgy, i.e., decreased salt consumption, alternate extraction techniques, etc.

I believe that this cost estimate shows the Tasminex project to be marginal. I hope this information will assist you in your 1973 Tasminex programs.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

TASMINEX PROJECT

SPECULATIVE COST ESTIMATE - SUMMARY

	A	B	C
Units WO_3 recovered per year	97,020	87,450	83,820
Overall WO_3 recovery	89%	80.3%	77%
Total Capital Required	$6,000,000	$6,000,000	$6,700,000

Processing Costs - $ per unit WO_3 recovered as synthetic scheelite (70% WO_3)

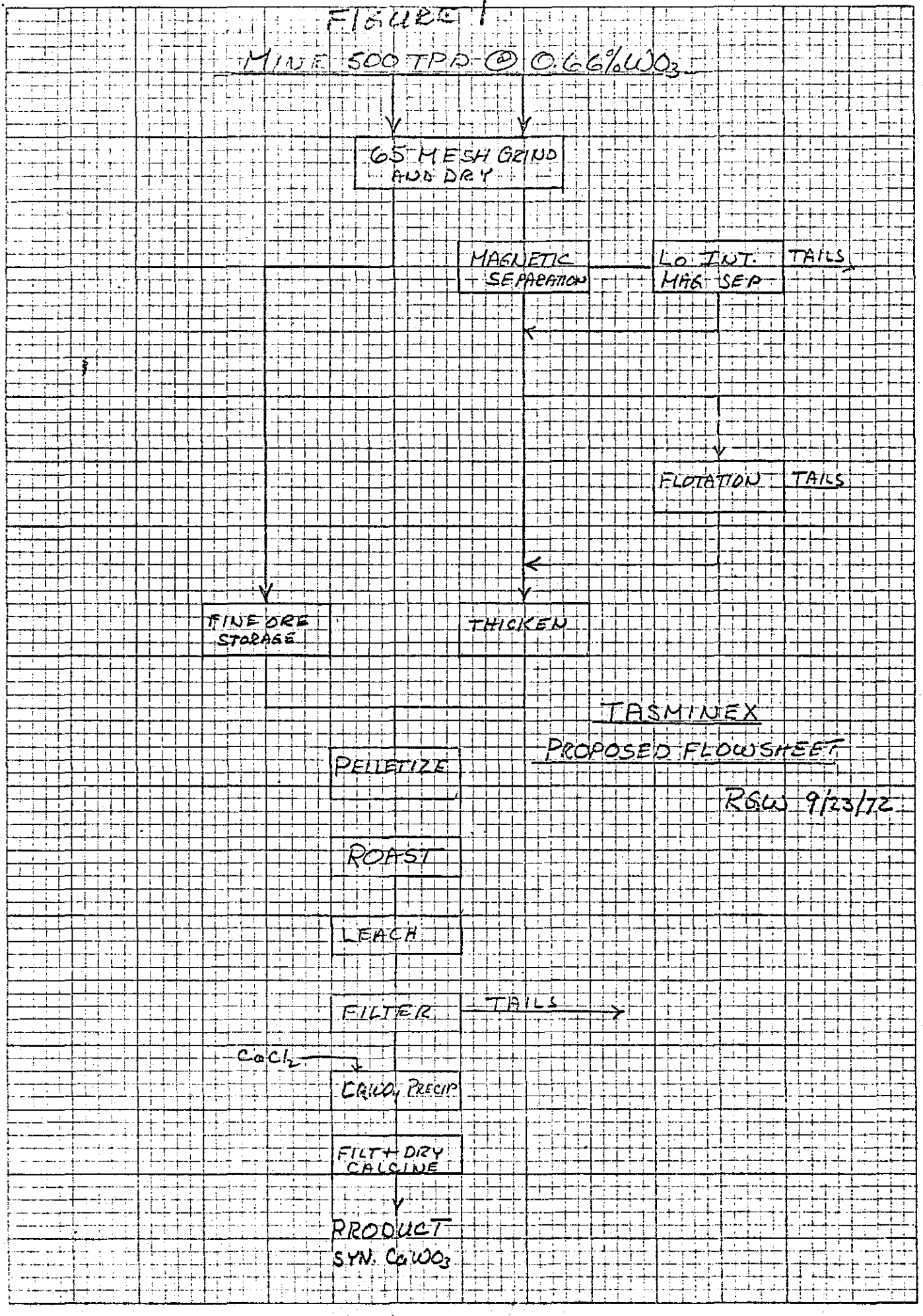
Mining Costs	6.12	6.79	7.09
Mill Labor Costs	3.83	4.25	4.65
Salt Roasting	13.06	9.22	7.77
Other Milling Costs	15.46	14.29	15.15
Depreciation	6.18	6.86	7.99
Total cost per unit recovered	44.65	41.41	42.63

% ROI @ Various WO_3 Sales Values (70% WO_3 - F.O.B. Tasmania)

@ $40.00 per unit WO_3	-	-	-
@ $45.00 " " "	0.6%	5.2%	3.0%
@ $50.00 " " "	8.6%	12.5%	9.2%
@ $55.00 " " "	16.7%	19.7%	15.5%

662017

FIGURE 1



TASMINEX

PROPOSED FLOWSHEET

RGW 9/23/72

TABLE I

TASMINEX PROJECT

ALTERNATE PROCESSES - MATERIAL BALANCE

	A(1)				B(1)				C(1)			
	Tons	Units	% Rec.	WO_3 Grade	Tons	Units	% Rec.	WO_3 Grade	Tons	Units	% Rec.	WO_3 Grade
Grind	500	330		0.76/ 0.56	500	330		0.76/ 0.56	500	330		0.76/ 0.56
Mag. Sep.	-	-	-	-	250/ 68	140/ 104	74	1.65	250/ 68	140/ 104	74	1.65
Flotation	-	-	-	-	-	-	-	-	68/ 6.5	104/ 89.5	86	15.0
Roast	500	330/ 300	91	0.66	318	294/ 270	91.5	0.93	257	282/ 259	92	1.13
Leach	-	300/ 294	98	-	-	270/ 265	98	-	-	259/ 254	98	-
Product	-	294/ 330	89	-	-	265/ 330	80.3	-	-	254/ 330	77	-

(1) Code:

A = Whole Ore Roast
B = 50% of Ore Upgraded by Magnetic Separation
C = 50% of Ore Upgraded by Magnetic Separation Plus Flotation

TABLE II

TASMINEX PROJECT

SPECULATIVE CAPITAL COST

	000's $ Per Year		
	A(1)	B(1)	C(1)
Grinding Circuit	1000	1000	1000
Magnetic Separation Circuit		250	250
Flotation Circuit			350
Pelletizing and Roast	1500	1100	1100
Chemical Plant	500	500	500
Finishing Plant	500	500	500
Total Equipment	3500	3350	3700
Services	800	800	1000
Eng. + Contingency	1000	1150	1250
Inventory + Working Cap.	700	700	750
Total Capital	6000	6000	6700
M Units Recovered per Year	97.02	87.45	83.82
Cost per Unit (10 yr. life)	$6.18	$6.86	$7.99

(1) Code:

A = Whole Ore Roast
B = 50% of Ore Upgraded by Magnetic Separation
C = 50% of Ore Upgraded by Magnetic Separation and Flotation

TABLE III

TASMINEX PROJECT

SPECULATIVE OPERATING COSTS

	000's $ Per Year		
MILLING COSTS	A(1)	B(1)	C(1)
Labor	372	372	390
Fuel and Power	500	300	300
Salt (Na_2CO_3 + NaCl)	1267	806	651
CaCl + Acid	50	50	50
Misc. Supplies	200	200	200
Maintenance Supplies	250	200	220
Aux. Expense	500	500	500
Total Milling Costs	3139	2428	2311
Units Recovered per Year	97.02	87.45	83.82
Cost per Unit WO_3 Recovered	32.35	27.76	27.57
MINING COSTS(2)			
@ $1.80 per ton & 1:1 strip ratio $3.60/ton ore x 165,000 =	594	594	594
Cost per unit WO_3 recovered	6.12	6.79	7.09

(1)Code:

A = Whole ore roast
B = 50% of ore upgraded by magnetic separation
C = 50% of ore upgraded by magnetic separation plus flotation

(2)Mining cost supplied by L. A. Wright

APPENDIX I

LABOR COSTS(1)

Category	No.	Yearly Cost
Supervisory	4	50,000
Operators	20	100,000
Shift Foreman	4	24,000
Maint. Foreman	4	24,000
Maint. Men	20	110,000
Instrument Men	2	12,000
Misc. Labor	4	20,000
Office	8	32,000
		372,000

SALT COSTS

Na_2CO_3 @ 160 lb./ton @ \$3.77/100 lb.	=	\$6.03/ton of Roast Feed
NaCl @ 250 lb./ton @ \$0.66/100 lb.	=	1.65/ton of Roast Feed
Total Salt Costs	=	\$7.68/ton of Roast Feed

(1) Operator wage rates reported to be \$50.00 weekly (\$60 U.S.) or ~\$3000/yr.

If accurate labor could be as low as \$250,000/year

Telex L.A. Wright from D. L. Mathias 9/11/72

b.c. E. E. Anderson
N. L. Grauerholz
D. J. Hansen

662022

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION — P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To: Mr. L. P. Twichell
Mining & Metals Division
Union Carbide Corporation
270 Park Ave.
New York, New York 10017

Date: September 14, 1972

Originating Dept.: Research and Development Department

Answering letter date:

Subject: Tasminex Project

cc to: Messrs. A. W. Hauck (2)
J. C. Stephenson
M. Stern
R & D File

Dear Lew:

Two more beneficiation memos have been completed and are attached for your information. The objective of this study was to compare the amenability of visually oxidized ore (magnetite at least surface oxidized) containing predominantly scheelite and unoxidized ore (magnetite essentially black) containing predominantly scheelite. The samples are subcomposites of our original Sample "C".

A summary of the test results is:

Product	% Wt. Distrib.	% WO_3 Assay	% WO_3 Distribution
Oxidized Ore			
Mag. conc.	67.1	0.61	33.1
Non-mag. conc.	33.9	2.40	65.9
Flot. tail	21.6	2.12	38.3
Flot. conc.	11.3	2.94	27.6
Calc. head	100.0	1.20	100.0
Unoxidized Ore			
Mag. conc.	74.3	0.32	17.7
Non-mag. conc.	25.7	4.33	82.3
Flot. tail	20.5	0.58	9.3
Flot. conc.	5.2	19.1	73.0
Calc. head	100.0	1.35	100.0

These data indicate that only the unoxidized portion of the ore body is amenable to magnetic separation and flotation. Thus, only the black ore has the potential of being upgraded economically.

Unfortunately, neither ore can be classed as representative of the ore body as both samples came from the same drill hole. (The bulk of the samples has been composited into Sample A, B, or C.) Furthermore, they can hardly be classed as typical because the WO_3 grade~1.3% WO_3 is considerably higher than the reported average ore grade.

However, when these results are reviewed in light of our previous studies, it seems reasonable to conclude:

1. The black or nearly black ores can be magnetically separated into a product containing >80% of the tungsten in a concentrate assay ~3 times the head analysis in a concentrate representing ~30% of the weight.

2. The scheelite in the Kara #1 ore can be floated into a rather high-grade concentrate provided the ore is relatively unoxidized.

3. The highly oxidized to moderately oxidized ore is not amenable to beneficiation and can only be economically recovered by salt roasting the whole ore.

In a separate study, the magnetic concentrate ores of varying degrees of oxidation were tested to determine if a cleaner step of low intensity magnetic separation would clean the magnetic tails. These results show increasing success with decreasing degree of oxidation. The results of this study show that a 0.27% magnetic waste product could be reduced to a 0.21% WO_3 waste with a concentrate assaying 4.28% WO_3 containing 24% of the total tungsten contained in the magnetic rougher product. Obviously, a flowsheet involving magnetic separation should include a low intensity magnetic cleaning stage.

This essentially concludes our laboratory upgrading program on Tasminex until such time as more "representative" samples are available. Some mineralogical work is still in progress and is expected to be completed soon. Upon completion of this work, a brief summary report will be written including a revised speculate estimate as to the probable economics based on these studies.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

Research & Development Department
P.O.Box 579, Niagara Falls, N. Y.
September 13, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

CC: Messrs. E. E. Anderson
W. B. DeAtley
J. S. Fox
D. J. Hansen
R & D File

FROM: N. L. Grauerholz

SUBJECT: Recovery of Tungsten Values From Tasminex Ore Samples
(Tungsten Ore/Raw Material Processing)

Summary

The magnetic concentrate from the three Categories A, B, and C was subjected to a dry, low-intensity hand magnet. The material with a low magnetic susceptibility was upgraded in WO_3 content, but on the more highly oxidized surface composites the tungsten loss in the magnetic fractions was still high.

Results and Conclusions

The magnetic fractions from flotation test work on the three Composites A, B, and C were subjected to a much lower magnetic field to produce a highly magnetic and weakly magnetic product. These results shown in Table I reveal that the tungsten has a tendency to concentrate in the weakly magnetic fractions. The weakly magnetic fraction from Category C is interesting because of the high tungsten content. Additional recovery by treating a magnetic fraction with weaker magnetic field strengths is one possible method of producing additional tungsten from the magnetic fraction of the ore body.

Examination of the tungsten in the weakly magnetic material revealed that most of the tungsten was present as nonscheelite. This material probably will not be amenable to additional upgrading by physical beneficiation but should be recovered by the developed roast-leach procedure.

N. L. Grauerholz

N. L. Grauerholz

NLG:rvg
Attach.

TABLE I

Tasminex Amenability Test
Magnetic Separation
Category A, B and C

Sample: Sala Magnetic Concentrate From Original Ores

Conditions: The magnetic concentrate from the Sala Laboratory Magnetic Concentrator was subjected to the magnetic field strength of a weak hand magnet. Material with a low magnetic susceptibility was thus rejected from the mass of the magnetite.

Results:

Category A - Sala Magnetic Concentrate

Product	% Wt.	% WO_3	Dist.WO_3
Highly Magnetic	97.5	0.46	91.7
Weakly Magnetic	2.5	1.62	8.3
Total Mags	100.0	0.49	100.0

Category B - Sala Magnetic Concentrate

Highly Magnetic	97.3	0.28	89.0
Weakly Magnetic	2.7	1.22	11.0
Total Mags	100.0	0.31	100.0

Category C - Sala Magnetic Concentrate

Highly Magnetic	98.5	0.21	76.3
Weakly Magnetic	1.5	4.28	23.7
Total Mags	100.0	0.27	100.0

662026

Research & Development Department
P.O.Box 579, Niagara Falls, N. Y.
September 12, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

CC: Messrs. E. E. Anderson
W. B. DeAtley
J. S. Fox
D. J. Hansen
R & D File

FROM: N. L. Grauerholz

SUBJECT: Recovery of Tungsten Values From Tasminex Ore Samples
(Tungsten Ore/Raw Material Processing)

Summary

Flotation tests on the oxidized and unoxidized subcomposites from Category C have shown the unoxidized subcomposite to be more amenable to flotation than the oxidized subcomposite.

Flotation of the unoxidized subcomposite, however, was not as good as expected and is believed to be due to a substantial fraction of nonscheelite tungsten. The concentrate grade and recovery from the two samples under similar conditions are:

	Cleaned Flotation Concentrate		
	% Wt.	% WO_3	Dist. WO_3
Oxidized Subcomposite	11.3	2.94	27.6
Unoxidized Subcomposite	5.2	19.1	73.0

Samples

The two subcategories from Category C used in compositing the oxidized and unoxidized fractions for flotation testing are shown in Table I. These subcategories were assembled to show the effect of oxidation on flotation grade and recovery. Unfortunately, the larger portions of the samples were originally combined into the three Categories A, B, and C. Only small fractions retained for mineralogical examination were available for the subcategories.

Processing

Combined magnetic separation-flotation tests were conducted on the subcategories from Composite C to determine the relationship between magnetite and its oxidation products-hematite and goethite and to the form of tungsten mineralization present.

Oxidized Subcomposite

A 1,000-gram test charge was ground to minus 100 mesh. The magnetic fraction was removed with the laboratory Sala Magnetic Separator and the nonmagnetic fraction was subjected to flotation. The conditions and results of this test are shown in Table II. These results show the magnetic iron fraction and flotation rougher tailing carries substantial tungsten. The presence of scheelite was detected in these fractions by U.V. light but not in the amount shown by assay. The losses were predominantly in the nonscheelite form. Two cleaning steps to increase tungsten grade did not produce an acceptable concentrate. Recovery was also poor.

Unoxidized Subcomposite

The unoxidized ore from Category C was blended into a composite for testing purposes. The procedure used was identical to the procedure used on the oxidized ore. The results, as shown in Table III, were different. Grade and recovery were raised. Grade improved from 2.94% WO_3 to 19.1% WO_3. Recovery improved from 27.6% to 73.0% of the total tungsten.

The largest loss of tungsten in the unoxidized ore was in the magnetic fraction where the tungsten losses were 17.7% of the total tungsten. The rougher tail still contained 0.57% WO_3, but represented a loss of only 7.2% of the total tungsten. A closely-panned concentrate from the rougher tails was examined under the U.V. light and revealed insufficient scheelite to account for this high tungsten loss.

Category C

The test results for Category C are shown in Table IV. These results are for comparison to the results shown for the oxidized and unoxidized subcategories. The concentrate grade at 28.6% WO_3 is higher than the 19.1% WO_3 concentrate shown for the unoxidized ore concentrate but recovery is less. The results on the unoxidized ore show a significantly better recovery.

Results and Conclusions

A comparison of the results from the tests for the oxidized and unoxidized subcategories from Category C show that the unoxidized ore has:

1. Lower WO_3 in magnetite.
2. Lower WO_3 in flotation rougher tailing.
3. Higher grade and recovery in flotation concentrate.
4. Higher pH values during test with same reagent quantities.

The losses in the magnetic fraction continue to appear to be related to the degree of oxidation of the magnetic fraction. The greater the oxidation the greater the tungsten loss. The unoxidized subcategory while significantly lower in goethite and hematite still contained some oxidized iron mineralization. Tungsten losses in the magnetic fraction were not lowered to a great degree by

finer grinding of the magnetic fraction or by attrition scrubbing or magnetic recleaning. The close association of tungsten with the magnetic fraction indicates that some of the nonscheelite tungsten are associated with minerals other than goethite and the earthy forms of hematite.

The tungsten losses in the flotation tailing fractions are extremely high. The distribution of this loss between scheelite and nonscheelite tungsten is not known but is believed to be mostly in the nonscheelite form. A previous sample, Drill Hole 102/3, produced a flotation rougher tailing of 0.14% WO_3 compared to the 0.57% WO_3 produced on the present unoxidized subcomposite from Composite C.

The pH of the natural ore appears to be a function of oxidation. The more highly oxidized ore has a more acidic pH. This trait might possibly be used in conjunction with color differences to determine the amenability of the ore to beneficiation. The iron oxides can vary widely in color within the same mineral species and in all cases may not be indicative of the degree of oxidation.

Additional test work is planned to determine the relationship of pH to beneficiation on a select group of some remaining individual drill hole samples.

N. L. Grauerholz

N. L. Grauerholz

NLG:rvg
Attach.

TABLE I

Tasminex Ore Samples
Subcomposites From Category C
Drill Hole Location

Oxidized Subcomposite C		Unoxidized Subcomposite C	
Drill Hole	Footage	Drill Hole	Footage
D112/8-14	22-43'	D112/37-41	97-114.5'
D112/28-36	28-97'	D115/27-43	77-124'
D115/22-26	63-77'	D113/48-50	205-220'
RTH1/2-12	3-32'	D118/19-21	50-57'
D109/19-21	56-65'	RTH1/13-18	32-56'

TABLE II

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category C - Oxidized Subcomposite

Sample: 1,000 Grams 8-Mesh Ore
OR-586

Conditions

	Time Min.	pH	Reagents, Lb./Ton of Original Feed			
			CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	17	8.0	0.5			
Cond.	15	8.1		1.0	1.0	0.5
R. Flot.	15					1.5 (staged)
Cl. Flot. No. 1	5	8.1			0.5	
Cl. Flot. No. 2	5	8.6			0.5	

Results

	% Wt.	% WO_3		Dist. WO_3
Mag Conc.	67.1	0.61		34.1
R. Tail	12.9	1.91		20.5
Cl. Tail No. 1	5.9	2.39	(1)	11.7
Cl. Tail No. 2	2.8	2.58		6.1
Cl. Conc.	11.3	2.94		27.6
Calc. Head	100.0	1.20		100.0
Actual Head		1.17		
Flotation Feed(1)	32.9	2.40		65.9

TABLE III

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category C - Unoxidized Subcomposite

Sample: 1,000 Grams 8-Mesh Ore
OR-586

Conditions

	Time Min.	pH	Reagents, Lb./Ton of Original Feed			
			CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	17	8.4	0.5			
Cond.	15	9.1		1.0	1.0	0.5
R. Flot.	15	9.1				1.5 (staged)
Cl. Flot. No. 1	5	8.9			0.5	
Cl. Flot. No. 2	5	8.6			0.5	

Results

	% Wt.	% WO_3		Dist. WO_3
Mag Conc.	74.3	0.32		17.7
R. Tail	17.1	0.57		7.2
Cl. Tail No. 1	2.8	0.64	(1)	1.3
Cl. Tail No. 2	0.6	1.61		0.8
Cl. Conc.	5.2	19.1		73.0
Calc. Head	100.0	1.35		100.0
Actual Head Flotation Feed(1)	25.7	4.33		82.3

* Analytical results not available

Ref. 1910-48

TABLE IV

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category C - Composite

Sample: 1,000 Grams 8-Mesh Ore
OR-586

Conditions

	Time Min.	pH	Reagents, Lb./Ton of Original Feed			
			CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	17		0.5			
Cond.	20	11.0	6.0			
Cond.	5	10.9		2.0	2.0	
R. Flot.	12	10.9				0.7
Cl. Flot. No. 1	5	10.7				0.06
Cl. Flot. No. 2	5	10.3	No Reagents			

Results

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	70.3	0.29	18.4
R. Tail	19.0	1.11	19.0
Cl. Tail No. 1	6.7	1.49	9.0
Cl. Tail No. 2	2.1	2.29	4.4
Cl. Conc.	1.9	28.60	49.2
Calc. Head	100.0	1.11	100.0
Actual Head		1.02	

File - Tasmania - Tasminex

662033

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

(Name) Mr. J. A. Straczek
vision Mining & Metals Division
cation Union Carbide Corporation
270 Park Avenue
New York, New York 10017

or to Messrs. A.W. Heuck/L.A. Wright

Date Sept. 11, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex Project
Analytical Procedure

Dear John:

During your recent visit we discussed with Hugh O'Hear the possibility of using the portable isotope analyzer as a tool in tungsten exploration. In particular, we asked Hugh to determine its potential with respect to the Tasminex tungsten ore. This investigation has been completed on 14 samples of ore and the results are contained in the attached memorandum.

I would conclude from these data that a P.I.A. unit would be most helpful in your exploration program in Tasmania because this instrument can determine tungsten content independent of the tungsten mineralogy. This, of course, is based solely on Kara #1 samples but it is reasonable that the curve would apply to the other ore locations with similar accuracy and, if necessary, a suitable curve could be made for each of the major tungsten occurrences in the region.

Should you have any further comments or questions on this subject, we will be glad to answer them.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

32

662034

Research and Development Department
P.O. Box 579, Niagara Falls, N.Y.
September 8, 1972

M E M O R A N D U M

TO: Mr. H. F. Wendt

COPIES: Messrs. N. L. Grauerholz
D. J. Hansen
C. M. Offenhauer
D. H. Traufler
R. G. Woolery
R. & D. File

FROM: H. J. O'Hear (3)

SUBJECT: Portable Isotope Analyzer - Tungsten
(General - Analytical Lab Reports)

A portable Isotope Analyzer has been used to determine control values in the Tempiute Pilot Plant operation (Niagara) and to analyze solutions from extraction studies of Bishop Organic. Chemical separations have also been used to extend the capability of the analyzer to a variety of diverse materials and analytical ranges by first eliminating interferences and then concentrating the tungsten values.

Interest has been expressed in use of the analyzer to determine tungsten content of various exploration samples and also to delineate ore samples of the Tasminex operation.

For exploration samples and ore bodies in general, the most efficient source-filter combination - and the one least subject to elemental interferences - uses Co-57 to excite the sample, with Tm and Er filters to isolate the resultant W ($K\alpha$) X-rays. Because of the high exciting energies involved, however, a relatively constant thickness of sample is required for the best analytical results. On the other hand, samples with particle sizes ranging up to approximately 8M can be tolerated.

While no specific conclusions can be drawn regarding exploration samples without actually evaluating similar specimens, previous work on the Tempiute Pilot Plant samples - where gross variations in matrix were encountered - indicates the possibility of collecting meaningful data. However, while analysis at low levels is possible, matrix effects can be severe and at low levels are always a source of uncertainty. The ultimate usefulness of the instrument in this application will undoubtedly relate directly to the experience of the operator.

Fourteen Tasminex head samples ranging in composition from 0.36% to 1.50% WO_3 were counted on the analyzer and the "counts" plotted against chemical values. Twenty gram samples were taken for the analyses and the "counts" represent the net counts between the two filters for 30 seconds, and are average values for 5 measurements. The results are plotted in Figure 1.

Eight Tasminex head samples identified as oxidized and unoxidized, (-100M) and (-8M) ore were also counted in the same manner and plotted against the chemical values of the corresponding analytical samples (-100M). The values are shown in Figure 2, where they are referenced to the calibration curve previously drawn (Figure 1). The one major variance from the curve could, and probably does, represent a valid chemical difference between the sample measured and the sample taken for chemical analysis.

As previously noted, sample thickness can influence the measured intensity when using Co-57 to excite the sample. The following table relates sample weight and thickness of -100M Tasminex Ore to variations in analyzer counts.

TABLE I

Sample Weight (g.)	Thickness (in.)	Net Counts/ 15 sec.
5	1/8	452
10	1/4	549
15	3/8	610
20	1/2	663
25	5/8	690
30	3/4	663

With analytical samples weighing in excess of 20 g. and with particle sizes of 8MxD, analyzer counts can be related to the tungsten contents of Tasminex Ore bodies within the precision noted. An increased precision is possible with longer counting times, if necessary.

While operation of the analyzer would probably be more reliable using the Tm and Er filters as indicated, it should be noted that, of several Tasminex samples tested, the matrices appeared to be sufficiently similar that a calibration curve could be drawn for use of the analyzer without filters.

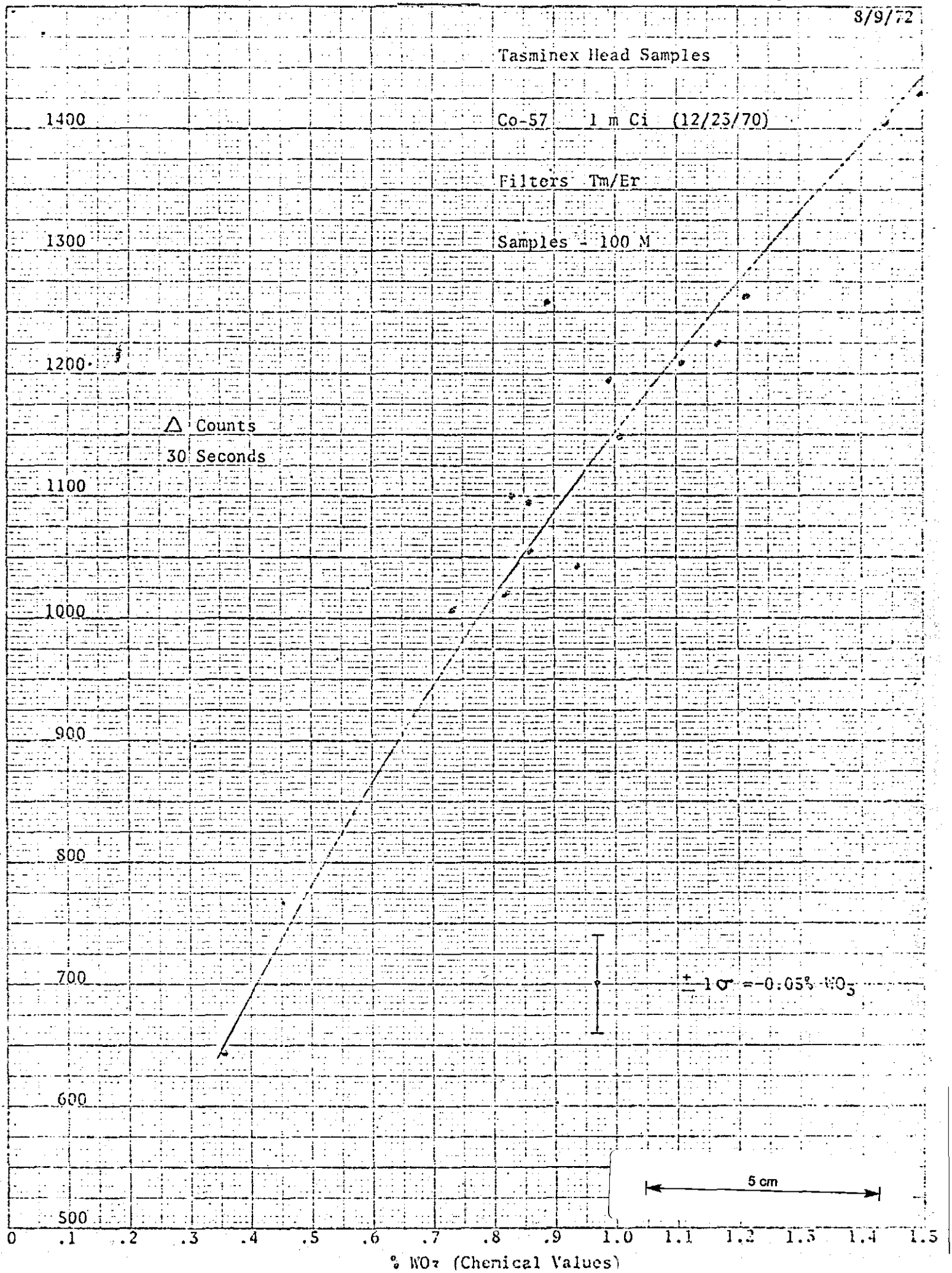
H. J. O'Hear

HJO:et
Attach.

FIGURE I

662036

8/9/72

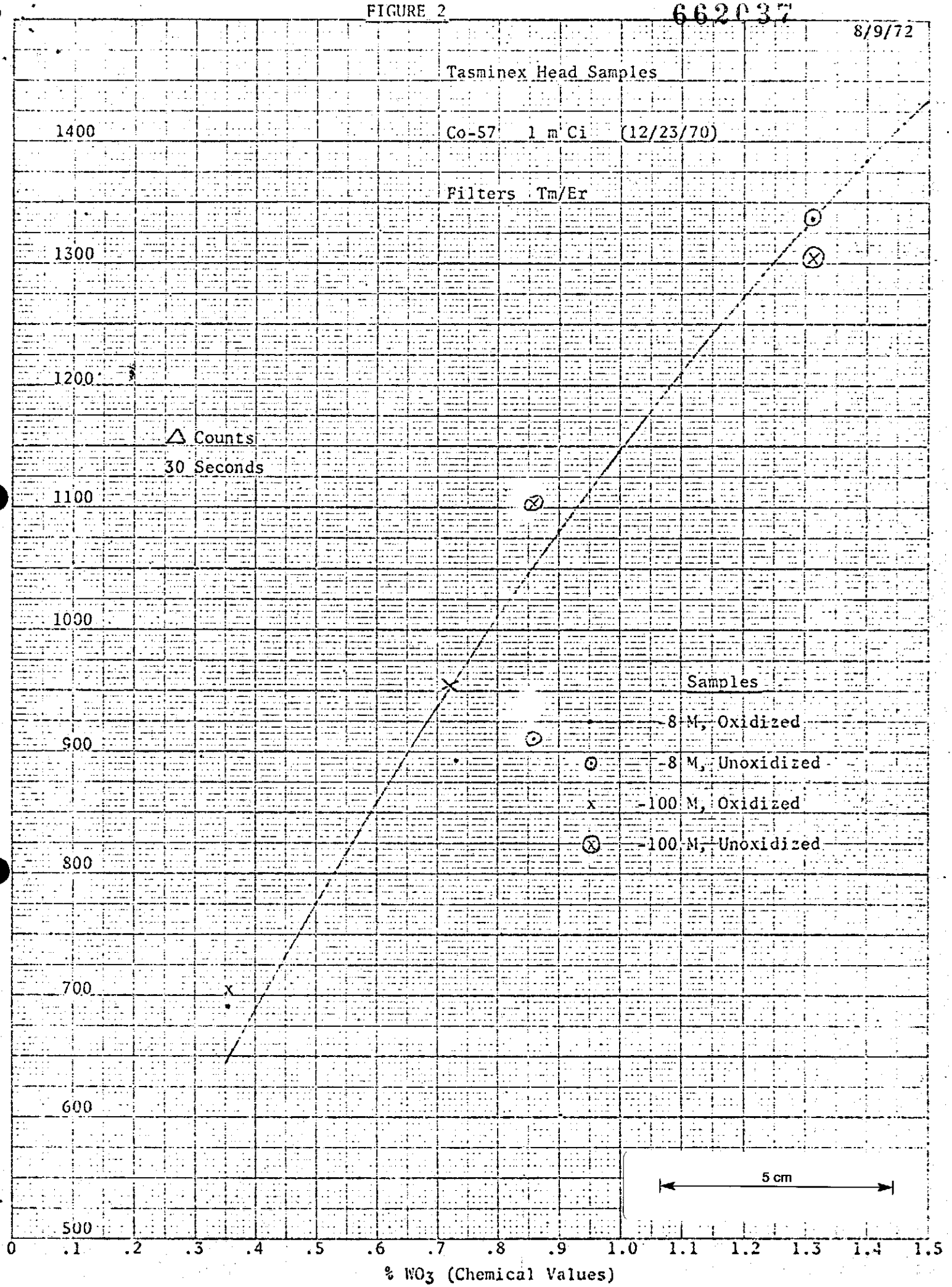


55

FIGURE 2

662037

8/9/72



662038

Research and Development Department
Niagara Falls, New York
August 30, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: Messrs. N. L. Grauerholz
D. J. Hansen
R. & D. File

FROM: R. Vedova

SUBJECT: Amenability Studies on Tasminex Tungsten Ore
(Tungsten Ore/Raw Material Processing)

Summary

Amenability studies on Categories A, B, and C indicate that:

1. Tungsten values present as scheelite can be easily recovered.
2. Nonscheelite tungsten values cannot be recovered by basic fatty acid or amine flotation practices, or by electrostatic methods.
3. WO_3 losses in the magnetic fraction can be reduced to ~15% with careful treatment.

Discussion

Preliminary amenability studies were carried out on the three Tasminex composites - Categories A, B, and C. Category A is an oxidized surface ore with little visible scheelite. Category B is an oxidized ore showing significant visible scheelite. Category C is basically an unoxidized ore having most of its tungsten values as scheelite.

Because Category C was primarily scheelite, it was investigated first using a basic fatty acid procedure. The magnetite was removed first, using a bar magnet, and then cleaned and washed several times. The nonmags were then subjected to an amine float to remove the hematite. This product was then cleaned, using Na_2SiO_3 to depress the scheelite. The fatty acid float was then carried out using a mixture of naphthenic acid and oleic acid (NFA) as a collector. Na_2CO_3 and Na_2SiO_3 were added as modifying agents. Na_2CO_3 and Na_2SiO_3 were also added to the two cleaning steps. The results are attached.

Product	% Wt.	% WO_3	% Dist.
WO_3 Concentrate	4.33	9.82	45.85
2nd Clr. Tail	2.79	1.12	3.37
1st Clr. Tail	1.78	1.98	3.80
Rougher Tail	13.20	1.30	18.50
Mags +400 M	36.62	0.18	7.11
Mags -400 M	34.70	0.20	7.48
Mag Scavenger	1.17	1.82	2.29
Fe_2O_3 Concentrate	0.71	1.26	0.97
Fe_2O_3 Clr. Tail	4.69	2.10	10.63

Although the WO_3 losses in the magnetite were kept relatively low, the rougher tail was high. Since no visible scheelite remained in the rougher tail, it must be assumed that this WO_3 was in the form of tungstite or some other non-flotable tungsten mineral.

Because of the high, nonscheelite tungsten losses, it was decided to test the effect of scavenging the rougher tails with amine type collectors. Three tests were run varying the conditions under which the different amines were tested.

The first test simply investigated the effect of different amines under the prevailing basic pH conditions. In the second test, the pH was depressed to the acidic side for flotation. Test 3 was carried out with a conditioning step before the amine flotation. Gallic acid, which complexes the WO_3 anion of a basic pH, was added to the conditioner. The results from these tests follow.

Test	Product	% Wt.	% WO_3	% Dist.
1	Conc. using Dodecylamine	5.6	1.54	6.6
	Conc. using Amine Sulfate	9.3	1.43	10.1
	Conc. using Amine Acetate	10.1	0.85	6.5
	Nonfloating Material	75.0	1.35	76.8
2	Conc. using Amine 220	59.6	1.55	70.6
	Conc. using Amine Sulfate	8.4	1.04	6.7
	Nonfloating Material	32.0	0.93	22.7
3	Conc. using Dodecyamine	31.1	0.6	16.1
	Conc. using Hydroxylamine Hydrochloride	35.5	1.15	35.4
	Nonfloating Material	33.4	1.68	48.5

Amine flotation was also carried out on Category B, but the results were much the same as with Category C.

Some testing was also carried out on the electrostatic separator, using Category A as the feed. Although some separation was noticed, particularly in the mags, it was not sufficient to warrant further testing along these lines. The results for the electrostatic unit were:

Product	% Wt.	% WO_3	% Dist.
Magnetic Conductors	3.02	0.38	1.18
Magnetic Nonconductors	1.51	0.96	1.49
Magnetic Conductors (Scavenged)	24.77	0.35	8.92
Magnetic Nonconductors (Scavenged)	1.02	0.97	1.02
Magnetic Middlings	29.36	0.44	13.2
Nonmag Conductors	0.57	1.74	1.02
Nonmag Nonconductors	0.88	1.97	1.79
Nonmag Conductors (Scavenged)	1.55	1.50	2.40
Nonmag Nonconductors (Scavenged)	1.28	2.88	3.80
Nonmag Middlings	4.77	2.15	10.54
Mags 400 M x D	12.44	0.56	7.16
Nonmags 400 M x D	18.81	2.45	47.39

At this times the ore is being re-categorized into more definite oxidized and unoxidized categories. No further work is being planned until those samples are available.

R. Vedova

RV:et

662041

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date August 14, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Mineralogy and Petrology of Tasminex Drill Holes

Copy to Messrs. A. W. Heuck ✓
J. C. Stephenson
M. Stern

Dear Lew:

Attached is an Interim Report on the Mineralogy and Petrology of Tasminex Drill Holes by Ed Anderson. At least in the samples sent to this laboratory, it is becoming evident that the bulk of the non-scheelite tungsten is present as a tungsten-bearing goethite. This has been supported by optical, X-ray, and microprobe studies. Additional probe work is in progress and we expect these results to confirm our current data.

The studies have also shown the intimate intergrowth of the goethite and hematite with the magnetite which accounts for the high magnetic tail we got from the magnetic separation of samples A, B, and C.

It has been speculated that a better classification of ore may be on the basis of color rather than scheelite estimation. For this reason we are further separating Sample C into oxidized and unoxidized ore, the former being yellow-brown and the latter being essentially black. Beneficiation studies will be conducted on each portion to determine amenability to upgrading. These results should be available in about one month.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

Research and Development Department
Mining and Metals Division - UCC
Niagara Falls, New York
August 7, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. W. B. DeAtley
J. S. Fox
N. L. Grauerholz
D. J. Hansen

FROM: E. E. Anderson

SUBJECT: Mineralogy and Petrology of Tasminex Drill Hole Samples - An Interim Report

SUMMARY

The drill hole samples received from the Kara property are being studied both in composites and as individual samples to produce mineralogic and geochemical characterizations adequate for definition of ore types and design of beneficiation circuits. To do this, it is highly desirable to incorporate into our study as much of the geologic data as are available and to develop an understanding of the geologic processes that have controlled ore emplacement and redistribution.

From interpretation of the samples and information received to date, it would appear that efficient beneficiation dictates a twofold classification. One ore class contains only scheelite as the ore mineral, the other class may contain some scheelite, but also has significant tungsten values in an as yet undefined, non-scheelite form. A method of discriminating these two classes in the field is suggested on the basis of color, ultraviolet fluorescence, and tungsten content.

Optical, X-ray, and microprobe studies indicate that the non-scheelite ore contains tungsten-bearing goethite (and possibly hematite). The intergrowth of this material with magnetite will make liberation difficult.

DISCUSSION

The crushed residues of the Tasminex drill hole samples referred to by R. T. Brandt in his letters of May 3, 1972 to McPhar Geophysics Pty. Ltd. and H. J. Henke were received at the Niagara Falls laboratory June 8, 1972. One quarter of each sample was split out for detailed mineralogic and related studies, while three quarters of each sample were combined for beneficiation test work, into the three categories indicated in Brandt's letters. The mineralogy and petrography of these composites are also being studied. The objectives of these

studies are to develop mineralogic, petrologic, and geochemical characterization of the Kara tungsten deposit in hopes of indicating the geologic processes that have affected ore emplacement and alteration and the distribution of primary (unoxidized) and secondary (oxidized) ore types. Such information should be useful not only in the development of beneficiation schemes for the various ore types, but also for estimation of ore reserves and to guide further exploration on the Kara and adjacent properties as well as similar deposits wherever they may be encountered.

To achieve these ends, five basic techniques are being employed. (1) The optical microscope is being used in both reflected and transmitted light modes for the qualitative and quantitative identification of the mineral phases present. (2) Similarly X-ray diffraction techniques are being used for more positive but less quantitative identifications. (3) The electron microprobe is being used to determine the distribution of elements in the various mineral species. This is particularly useful in charting the location of the tungsten in variously oxidized samples. (4) Separation of the various mineral constituents by magnetic, gravity, and hand-picking methods is also in progress. These mineral concentrates facilitate phase identification, and chemical analyses of the various fractions indicate which constituents contain the tungsten values. (5) Both quantitative and semiquantitative chemical analyses of sample sets also indicate changes related to the geologic processes that have affected tungsten distribution and migration.

To date, our investigations have concentrated on optical, X-ray, and chemical analyses of the three composite "head" samples and various magnetic separates of these, and X-ray and chemical (spectrographic) analyses of the individual samples from drill hole D115. This drill hole suite was chosen for detailed study because it appears to be the most completely sampled hole received in Niagara Falls.

The optical examination, particularly of samples containing the finely crystalline (earthy) goethite, was delayed while techniques for impregnation and polishing of this material were developed. The normal sample preparation methods used in the Niagara Falls Metallographic laboratory did not produce the flat surfaces necessary for either optical examinations or microprobe analyses of intergrowths of the earthy goethite and the other constituents. Our present technique for the preparation of these recalcitrant subjects is still not optimum but did well enough to permit the photomicrographs in this report. These sections have not yet been examined on the microprobe to verify the non-scheelite occurrence of the tungsten in the oxidized material. As can be seen in Figures 1-6, 9 and 10, goethite has replaced magnetite-hematite intergrowths in material from all three category composites. Incipient alteration of magnetite to hematite commonly along crystallographic directions is present in even the least oxidized specimens examined (Figures 7-9). In contrast, goethite is very minor in the least oxidized samples. Here I am referring not to all of the composite C material, but to the portions of it that are least oxidized and core samples previously received (B102/3).

Table I shows the approximate mineralogic composition of composites of categories A, B, and C material. Actual values are probably within 5 or 10% of those reported. As indicated in the table, magnetite (Fe_3O_4), hematite (Fe_2O_3), and goethite ($HFeO_2$) are the principal constituents, with minor but significant amounts of scheelite ($CaWO_3$), vesuvianite $[Ca_{10}(Mg,Fe)_2Al_4Si_9O_{34}(OH)_4]$, and diopside ($CaMgSi_2O_6$) being noted in composite C. Trace amounts of scheelite were noted in composites A and B by ultraviolet fluorescence (see Tables II and III), but were not sufficiently abundant for detection by X-ray diffraction in the head samples. Quantitative analyses for ferrous and ferric iron as well as several other elements, and semiquantitative spectrographic analyses for the three composites are given by N. L. Grauerholz in his memorandum to R. G. Woolery dated July 20, 1972. If we assume that all of the ferrous iron in the three composites is in the form of magnetite (this is a valid approximation), the analyses for the composite head samples indicate enough ferrous iron for 33% magnetite in A, 21% in B, and 46% in C. The remaining ferric iron would be present as hematite and goethite. These magnetite values appear to be somewhat low in comparison with that indicated by X-ray diffraction (Table I). When time permits, optical modal analyses will also be made for comparison. In any event, both methods indicate that composite B contains less magnetite than composite A. Or simply more of the iron in B is oxidized than that in A.

It would appear that the breakdown of scheelite is not entirely coincident with the alteration of magnetite to goethite since scheelite is decidedly more abundant in B than in A, although the iron in B is more completely oxidized than in A. The question remains, does all of the goethite contain tungsten? A preliminary microprobe analysis indicates that tungsten in iron mineral(s) from the B1 pit sample ranges up to about 25%. The identification of the iron minerals was uncertain due to the poor polish of the sections. For the same reason, tungsten values are questionable. It is hoped that study of new sections of the three composite samples will soon resolve these problems.

Preliminary flotation results of composite C material by N. L. Grauerholz (re letter of 7/20/72 to R. G. Woolery) indicate that on the order of 30-40% of the tungsten in this composite is not recoverable by a scheme designed to float scheelite. This would seem to indicate that composite C as presently defined contains significant values in a form that is not scheelite. To date, all test work to physically concentrate the tungsten in the composite C sample has resulted in prohibitive losses of tungsten in tailings products. This is in contrast to previous work on truly unoxidized or only incipiently oxidized samples from the Kara deposit which appeared to be quite amenable to magnetic, gravity, and flotation techniques.

That the category C constituent samples contain various proportions of the iron oxide minerals magnetite, hematite, and goethite is also evident from their colors. Tables II and III are logs of all samples received in Niagara Falls from drill holes D112 and D115. Listed in the log for each sample are color designations according to the Rock-Color Chart distributed by the Geological

Society of America, and relative scheelite abundance as determined by visual estimate under ultraviolet illumination. The other drill hole samples available in Niagara Falls are currently being logged in a similar manner. On the basis of color, the category C samples have been segregated into those that are relatively oxidized and those that are relatively unoxidized. For the purpose of this classification, a distinction was made between the lighter tan, brown, or yellow colors and the darker brown to black colors. The colors of the more oxidized samples include light brown, 5 YR 5/6, and moderate brown, 5 YR 4/4. Colors of the less oxidized samples include moderate brown, 5 YR 3/4, grayish brown, 5 YR 3/2, dusky yellowish brown, 10 YR 2/2, and dark yellowish brown 10 YR 3/2 and 10 YR 3/2. The lighter brown to yellow colors are usually due to the presence of goethite. This has been checked by X-ray diffraction for drill holes D112 and D115. A sharp decrease in goethite and an increase in magnetite occur between samples 36 and 37 in hole D112. An increase in magnetite and a decrease in hematite occur between samples 20 and 21 in hole D115. Goethite does not diminish markedly in hole D115 until sample #24.

I find that the picking of a boundary between relatively oxidized and unoxidized zones is a subjective thing and can best be accomplished by a combination of observing the color of the dry crushed rock, the amount of fluorescent scheelite, and to some extent depth of sample and sequence of colors down the hole or a distinct color change. The strongest indicators appear to be the color change in combination with a high scheelite to tungsten ratio. The problem here is that the visual estimation of scheelite, at least in the crushed samples we have received, is highly subjective.

At this writing I am still not absolutely positive as to the nature of the form of the secondary tungsten occurrence. I still favor the idea that some or all of the goethite and possibly some of the hematite are carrying the tungsten in the oxidized zone. This should be resolvable by the impending microprobe work. If we assume that goethite is the secondary tungsten mineral, we must examine the problem of liberating it from the magnetite before magnetic separation. From Figures 1-6, 9 and 10 we can see that the liberation of the secondary tungsten mineral (goethite) from the magnetite may require extremely fine grinding.

It would appear that only the strictly unoxidized ore will be amenable to significant physical upgrading (other than removal of magnetite). This will require discrimination between ore with scheelite as the only ore mineral from that containing significant amounts of non-scheelite ore.

From these textures and from the early work of N. L. Grauerholz on magnetite removal, it would appear that the magnet needs to be adjusted to remove only the highly magnetic particles. That is, only the liberated magnetite. It would be far better to have some magnetite in the roast feed than to lose much tungsten in the magnetic product.

E. E. Anderson

EEA:ms

TABLE I

Mineralogic Composition of Tasminex Composites
by X-Ray Diffraction of the Head Samples

Composite Category	Per Cent		
	Magnetite	Hematite	Goethite
A	55	35	10
B	45	45	10
C*	70	25	5

*Per cent magnetite in "C" is by difference and includes less than 5% each of scheelite, vesuvianite, and diopside (?).

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TABLE II

Hole D 112 Log

Sample No.	Depth, ft.	N.F. N.B. No.	% W	Scheelite Abundance	Color
1-3	0-9	Not Received			
4	9-12	1903-9-77	0.61	Essentially no scheelite	Moderate brown 5 YR 4/4
5	12-16	-78	0.43	" " "	" " "
6	16-19	-79	0.74	" " "	" " "
7	19-22	Not Received	0.71		
8	22-25	-81	1.47	Moderate scheelite	" " "
9	25-28	-82	0.33	Sparse scheelite	Moderate brown 5 YR 3/4
10	28-31	-83	0.68	Moderate scheelite	" " "
11	31-34	-84	0.87	" "	" " "
12	34-37	-85	0.86	" "	Moderate brown 5 YR 4/4 - 5 YR 3/4
13	37-40	-86	0.59	" "	" " " "
14	40-43	-87	0.50	" "	" " " "
15-27		Not Received			
28	78-81	-88	0.72	Sparse scheelite	Moderate brown 5 YR 3/4
29	81-84	-89	0.88	" "	Moderate brown 5 YR 4/4 - 5 YR 3/4
30	84-87	-90	0.81	" "	Moderate brown 5 YR 3/4
35	87-89	-91	0.52	" "	Grayish brown 5 YR 3/2
31	89-91	-92	0.65	" "	" " "
32	91-93	-93	0.69	Sparse - moderate scheelite	" " "
33	Not Listed	-165		Very sparse scheelite	Moderate brown 5 YR 4/4
34	Not Listed	-166		Sparse scheelite	Moderate-light brown 5 YR 4/4 - 5 YR 5/6
36	93-97	-94	1.22	Moderate scheelite	" " " " "
37	97-100	-95	1.39	Abundant scheelite	Moderate brown 5 YR 3/4
38	100-103	-96	0.27	No scheelite	Pale brown 5 YR 5/2
39	103-107	-97	0.11	Sparse scheelite	Dusky brown - dusky yellowish brown 5 YR 2/2 - 10 YR 2/2
40	107-111.5	-98	0.78	Abundant scheelite	Grayish brown 5 YR 3/2
41	111.5-114.5	-99	0.7[illegible]	Abundant scheelite	Moderate brown 5 YR 4/4 - 5 YR 3/4

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TABLE III

Hole D 115 Log

Sample No.	Depth, ft.	N.F. N.B. No.	% W	Scheelite Abundance	Color
1-4	0-12	Not Received			
5	12-15	1903-9-103	0.47	Essentially no visible scheelite	Moderate brown 5 YR 4/4
6	15-18	-104	0.57	" " "	" " "
7	18-21	-105	0.74	" " "	" " "
8	21-24	-106	1.61	" " "	" " "
9	24-26	-107	1.14	Very little scheelite, one or two grains	Moderate brown 5 YR 4/4 - 5 YR 3/4
10	27-30	-108	0.94	" " " "	" " " "
11	30-33	-109	0.64	" " " "	" " " "
12	33-36	-110	0.43	Obvious increase in scheelite from above	" " " "
13	35-39	-111	0.51	About like 12	" " " "
14	39-42	-112	0.70	" " "	" " " "
15	42-45	-113	0.81	" " "	" " " "
16	45-48	-114	0.88	Somewhat less scheelite than 12-15	Moderate brown Predominantly 5 YR 4/4 " "
17	48-51	-115	0.73	About like 16	" " " "
18	51-54	-116	0.72	" " "	" " " "
19	54-57	-117	0.48	" " "	" " " "
20	57-60	-118	0.54	Slightly less scheelite than 16-19	" " " "
21	60-63	-119	0.49	Somewhat more scheelite than 20, like 16-19	" " " "
22	63-66	-120	0.47	About like 21. Slightly darker brn. than 23-26	Moderate brown 5 YR 3/4

TABLE III (Continued-2)

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Sample No.	Depth, ft.	N.F. N.B. No.	% W	Scheelite Abundance	Color
23	66-69	1903-9-121	0.55	Scheelite definitely more abundant - moderate	Moderate brown 5 YR 3/4
24	69-72	-122	0.68	Very slightly more scheelite than 23	" " "
25	72-75	-123	0.68	About like 24	" " "
26	75-77	-124	0.63	" " "	" " "
27	77-79	-125	0.47	Abundant scheelite	Dusky yellowish brown 10 YR 2/2
28	79-81	-126	0.67	Abundant scheelite	Dusky yellowish brown 10 YR 2/2 - 5 YR 2/1 to brownish black
29	81-84	-127	0.22	Much less scheelite than 27 & 28, about like 23	Dark yellowish brown 10 YR 4/2
30	84-87	-128	0.76	Abundant scheelite	" " " 10 YR 3/2
31	87-90	-129	0.91	" "	" " " 10 YR 4/2
32	90-93	-130	0.24?	" "	" " " "
33	93-96	-131	0.27?	Abundant scheelite, megascopic pyrite intergrown with magnetite & transparent silicate	10 YR 3/2
34	96-99	-132	1.23	More scheelite than 30-33	Dark yellowish brown 10 YR 4/2
35	99-102	-133	3.12	" " " 34	" " " "
36	102-105	-134	0.42	Much less scheelite than 34 & 35 but moderately abundant	" " " "
37	105-108	-135	0.21	" " " "	" " " "
38	108-111	-136	0.41	" " " "	" " " "
39	111-114	-137	0.26	" " " "	" " " "
40	114-117	-138	0.46	" " " "	" " " "
41	117-120	-139	0.47	" " " "	" " " "
42	120-122	-140	0.84	" " " "	Grayish olive - dark brown 10 YR 4/2 - 10 YR 4/2
43	122-124	-141	0.47	" " " "	Moderate yellowish brown 10 YR 5/4

INTERNAL CORRESPONDENCE

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AUG 2 1972

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date August 1, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex Project
Evaluation of Samples A, B, & C

Copy to Messrs. A. W. Heuck (2)
J. C. Stephenson
M. Stern

Dear Lew:

Attached are the laboratory reports pertaining to the physical upgrading and salt roast amenability tests conducted on Samples A, B, and C as submitted by UCEX. The samples represent theoretical degrees of oxidation from Sample A which contains very little scheelite to Sample C which is predominantly scheelite but not necessarily unoxidized ore.

The results showed that Sample C is amenable to magnetic separation, Sample B is probably satisfactory but questionable, and Sample A would probably lose too much WO_3 in the tails to be economically attractive. We have arbitrarily set 0.30% WO_3 as a cut-off point for tails as this is our "best guess" now as to what level of tungsten can be economically treated by salt roast.

Both gravity and flotation methods were tried on all three samples and it is apparent that Samples A and B would not be amenable to beneficiation. Sample C showed promise by both gravity and flotation to the extent that the scheelite responded quite favorably. However, the nonscheelite content was sufficiently high in this particular sample that overall selectivity, particularly recovery, would not permit either practice on an ore as represented by Sample C.

Salt roast amenability tests showed that this means of recovery is good for all samples and is virtually independent as to the form of WO_3. Tungsten recoveries of 91-94% were obtained on all samples and could probably be improved with additional optimization studies. Salt roast chemical costs are estimated at $3.30 per unit of WO_3 for feeds of 1.0% WO_3 or greater. There is some indication, however, that the cost may escalate significantly at grades lower than 1.0% WO_3.

This study has shown the significant influence of degree of oxidation on the physical beneficiation of the ore. Further, it appears that every effort should be made to ensure at least a 1.0% WO_3 feed to the mill. Our current program will be directed at examining unoxidized ore as to its amenability to upgrading techniques. We expect to separate the oxidized and unoxidized portions from Sample C and study these independently. These two fractions will be selected on

Unless such a separation is practical from a field or mining standpoint isn't this approach a waste of time?

a basis of observation as to the oxidation state of the predominate iron minerals. The black pulp samples will be combined as the unoxidized ore and the brown-reddish pulps will be combined as the oxidized portion of Sample C.

The next laboratory report will be issued upon the completion of the above study.

Very truly yours,

R. G. Woolery

RGW/bsn

Research and Development Department
Mining and Metals Division - UCC
Niagara Falls, New York
July 26, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. E. E. Anderson
W. B. DeAtley
N. L. Grauerholz
D. J. Hansen

FROM: J. S. Fox

SUBJECT: Recovery of Tungsten Values from Tasminex Ore Samples

SUMMARY

Samples of three types of ore were received from the Tasminex ore deposit. Category A contained dominantly non-scheelite tungsten ore, Category B contained a mixture of scheelite and non-scheelite ore, while Category C contained dominantly scheelite ore. Amenability tests were made on splits from each type of ore using -8 mesh materials or -100 mesh materials for the tests.

These ore samples were magnetically treated by the Minerals Beneficiation Group under the supervision of Mr. N. L. Grauerholz. The upgraded concentrates from each ore type were then tested for their amenability toward the Na_2CO_3-NaCl roast-water leach process. These tests were made on the -8 mesh and the -100 mesh concentrates.

The results obtained showed that all of the ores tested (0.71% to 1.10% WO_3) were amenable to the Na_2CO_3-NaCl roast-water leach process. Using 8% Na_2CO_3 and 12% NaCl in the 1-hour roasts at 900°C., from 92% to 93% of the tungsten values was solubilized from the -100 mesh materials. Slightly lower solubilizations of 88 to 89% WO_3 were obtained when -8 mesh materials were employed.

When the concentrates from each ore type were tested, it was found that good tungsten solubilizations of 91% to 94% were obtained from the concentrates produced from the three ore types. It appears that it is not necessary to further grind the -8 mesh concentrates obtained from either Category A or B since the -100 mesh materials were no more amenable to the Na_2CO_3-NaCl roast-water leach process than the -8 mesh materials. With Category C, better results were obtained on the -100 mesh materials. These results were obtained in 1-hour roasts at 900°C. using 12% to 18% Na_2CO_3 along with 24% to 36% NaCl. The chemical costs for roasting these concentrates (1.22% to 2.79% WO_3) ranged from 17 to 30¢ per pound of WO_3 solubilized from the 1-hour roast at 900°C.

Based on all of the experimental data obtained to date, the following chemical costs have been estimated based on direct roasting of Categories A and B and roasting the concentrate from Category C.

% Ore Grade	Yield After Roast	Na_2CO_3 Usage #/Unit WO_3 Ext.	Costs/Unit WO_3 at $35 per Ton	NaCl Usage #/Unit WO_3 Ext.	Costs/Unit WO_3 at $7.10/Ton	Ppt. Costs $/Unit WO_3 at 3¢ per # WO_3	Total Cost $/Unit WO_3
0.7	93	244	4.27	364	1.29	0.60	6.16
1.0	92	164	2.87	144	0.51	0.60	3.98
Conc. Grade %							
2.5	93	146	2.56	196	0.70	0.60	3.86

Further tests may show that this cost may be slightly reduced since the minimum amount of Na_2CO_3 or NaCl has not been established.

DISCUSSION

Since the last memorandum on July 7, 1972, experimental work was continued on the recovery of tungsten values from the oven-dried non-magnetic portion of a tungsten composite of surface and subsurface ore using a constant amount of NaCl and varying amounts of Na_2CO_3 in the roast. This sample was prepared by the Minerals Beneficiation Group of our Laboratory under the supervision of Mr. N. L. Grauerholz. In addition, amenability tests were made on three groups of ore samples which were recently received from Tasminex. These three ore samples were labeled (1) Category A - which is dominantly a non-scheelite ore, (2) Category B - which is a mixed ore of scheelite and non-scheelite, and (3) Category C - which is dominantly scheelite. These samples were divided in the described groups by Mr. R. T. Brant, Regional Geologist from the Australia and New Zealand Exploration Company. Each group was split and ground to give a -8 mesh ore sample and a -100 mesh sample.

A magnetic separation was made on portions of each of the described six samples by the Minerals Beneficiation Group. Amenability tests were then made on each of these six magnetically upgraded samples.

1. Effect of a Constant Amount of NaCl and Varying Amounts of Na_2CO_3 in the Roast on Amount of Tungsten Solubilization

In the previous memorandum of July 7, 1972, excellent tungsten solubilization was obtained on a Tasminex concentrate (2.43% WO_3) by roasting for 1 hour at 900°C. using 36% NaCl and 8% or 12% Na_2CO_3 (g./100 g. concentrate). A test was made as described except the Na_2CO_3 was reduced to 5%. The conditions and results of these tests are shown in Table I. These results show that poor tungsten solubilization occurs (approximately 2%) when the Na_2CO_3 usage is too low. These

results also show that the pH of the aqueous leach solution drops from 11.7 to 6.1 by reducing the Na_2CO_3 from 8% to 5%. Thus, it appears that for this concentrate, more than 5% Na_2CO_3 is required but not more than 8% Na_2CO_3 along with no more than 36% NaCl is required to solubilize 97% of the tungsten values by the Na_2CO_3-NaCl roast process.

2. Amenability Tests on Tasminex Ore Samples

Six samples of Tasminex ore samples were received from the Minerals Beneficiation Group for Na_2CO_3-NaCl roast amenability tests to solubilize the tungsten values by a subsequent water leach. As previously mentioned, these ore samples were divided into three categories as follows: Category A which is dominantly a non-scheelite ore, Category B which is a mixture of scheelite and non-scheelite ore, and Category C which is dominantly a scheelite ore. Each group was ground to -8 mesh and sampled. The chemical analyses of these samples are shown in Table II and the spectrographic analyses in Table III. A screen analysis of these samples is shown in Table IV. Samples of each of the -8 mesh ore groups were ground to -100 mesh.

Each of the above six samples was roasted for 1 hour at 900°C. with 8% Na_2CO_3 along with either 12% or 24% NaCl. These conditions were employed since they gave the best results in previous tests on other Tasminex concentrates. The conditions and results of these tests are shown in Table V. The following results show:

Category A

Increasing the NaCl from 12% to 24% in the 8% Na_2CO_3 roasts shows very little increase in tungsten solubilization (less than 1%).

A comparison of tungsten solubilization of the -8 mesh ore with the -100 mesh ore showed that a small increase of approximately 3% was obtained by grinding the ore to -100 mesh.

The above results show that good solubilization of 92% was obtained from Category A using 8% Na_2CO_3 along with 12% NaCl in the roast. To achieve these extractions, 8.2 pounds of Na_2CO_3 and 12.2 pounds of NaCl were employed per pound of WO_3 solubilized. The chemical costs for these reagents based on $35.00 per ton for Na_2CO_3 and $7.10 per ton for NaCl would be approximately 19¢ per pound of WO_3 solubilized.

Category B

These tests show that increasing the NaCl from 12% to 24% in roasts containing 8% Na_2CO_3 increased the tungsten solubilization very little. Increasing the mesh size from -8 mesh to -100 mesh increased the tungsten solubilization from 89% to 93%. These tests show that good tungsten solubilization was obtained on this low-grade ore (0.71% WO_3) using 12.2 pounds of Na_2CO_3 and 18.2 pounds of NaCl per pound of WO_3 solubilized. These chemical costs would be about 28¢ per pound of WO_3 solubilized.

Category C

Increasing the NaCl from 12% to 24% in the 8% Na_2CO_3 roasts shows very little increase (approximately 1%) in tungsten solubilization of the -100 mesh samples. Increasing the mesh size from -8 mesh to -100 mesh increased the WO_3 solubilization approximately 3% when 8% Na_2CO_3 with 12% NaCl was employed in the roast. These results show that good tungsten solubilization was obtained using 8.9 pounds of Na_2CO_3 and 13.4 pounds of NaCl per pound of WO_3 solubilized. These chemical costs would be approximately 20¢ per pound of WO_3 solubilized.

3. Amenability Tests on Tasminex Non-Magnetic Concentrates

Magnetic separations were made on each of the three category samples using -8 mesh samples in one case and -100 mesh in the other. These six samples, upgraded by the Minerals Beneficiation Group, were given amenability tests. In these tests, 8% to 18% Na_2CO_3 was employed along with 24% to 36% NaCl in the roasts which were made with a 1-hour retention time at 900°C. The conditions and results of these tests are shown in Table VI.

Category A

In the roasts of the non-magnetic fraction of Category A, increasing the mesh size of the ore from -8 mesh to -100 mesh did not increase the tungsten solubilization when 12% Na_2CO_3 and 36% NaCl were employed in the roast. In each case good tungsten solubilization (94%) was obtained. Increasing the amount of Na_2CO_3 to 16% along with 36% NaCl increased the tungsten solubilization to 95.7% when the -8 mesh material was roasted. When the amount of Na_2CO_3 was reduced to 8% in the above roast with 36% NaCl, the tungsten solubilization dropped sharply to 56%. The pH of the leach liquor also dropped sharply to 7.0 compared to a pH of 11.1 when 12% Na_2CO_3 and 36% NaCl were employed in the roast. To achieve 94% WO_3 extraction (Test 2, Table VI), 5.9 pounds of Na_2CO_3 and 17.8 pounds of NaCl were employed. These chemical costs would be approximately 17¢ per pound of WO_3 extracted based on Na_2CO_3 at $35.00 per ton and NaCl at $7.10 per ton.

Category B

In the roast of the -8 mesh material, good tungsten solubilization (94%) was obtained on the -8 mesh concentrate when 12% Na_2CO_3 along with 36% NaCl was employed. When the amount of Na_2CO_3 was decreased to 8%, the tungsten solubilization decreased (78%). No increase in tungsten solubilization was obtained by roasting the -100 mesh ore with 12% Na_2CO_3 containing 36% NaCl (compare Tests 7 and 9 in Table VI). To obtain the 94% WO_3 solubilization, 10.5 pounds of Na_2CO_3 and 32 pounds of NaCl were employed per pound of WO_3 extracted. These chemical costs would be about 30¢ per pound of WO_3 solubilized. Tests with 12% Na_2CO_3 and lower amounts of NaCl in the roast should be made if lower costs are required at this stage of development.

Category C

In these tests, a fairly good extraction of 93% was obtained using 18% Na_2CO_3 and 24% NaCl in the roast. With lower amounts of Na_2CO_3, a decrease in tungsten solubilization was obtained in roasts of the -100 mesh material. When 8% Na_2CO_3 and 36% NaCl were employed in a roast of the -100 mesh material, only 12% WO_3 was solubilized and the pH dropped to 6.8.

To obtain 93% WO_3 solubilization, it required 7.3 pounds of Na_2CO_3 and 9.8 pounds of NaCl per pound of WO_3 extracted. These chemical costs would be about 17¢ per pound of WO_3 solubilized.

In all cases so far, it was noted that when the pH of the leach solution is low (below 10), a decrease in tungsten solubilization occurs. This may indicate when a deficiency in the amount of Na_2CO_3 occurs in the roast.

J. S. Fox

JSF:ms

TABLE I

Effect of a Constant Amount of NaCl
with Various Additions of Na_2CO_3 in Roasts
of Tasminex Ore Concentrate
on Tungsten Solubilization in Subsequent Water Leach

Roast Conditions: (*Non-Mag fraction (2.43% WO_3)
(900°C.
(1 hr.
(36% NaCl (36 g./100-g. Sample)

Test No.	Na_2CO_3 Added % or g./100-g. Sample	% WO_3 Extracted	WO_3 in Tails %	pH of Filtrate	WO_3 Material Balance
1	12.0	97	0.08	10.9	83
2	8.0	97	0.08	11.7	95
3	5.0	1.6	1.91	6.1	97

*Non-magnetic fraction of surface and subsurface Tasminex ore composite

TABLE II

Chemical Analyses
of Various Tasminex Ore Samples

(Supplied by Minerals Beneficiation Group)

Sought	Found in Per Cent		
	Category A Dominantly Non-Scheelite	Category B Mixed Scheelite and Non-Scheelite	Category C Dominantly Scheelite
WO_3*	1.05-1.07	0.68-0.71	0.97-1.10
SiO_2	4.48	4.03	9.29
Fe^{++}	7.82	5.19	10.56
Fe^{+++}	52.5	55.57	45.96
Ca	0.06	0.13	1.99
Al	1.71	1.68	1.67
Cu	0.068	0.054	0.063
Pb	0.056	0.037	0.013
Zn	0.067	0.075	0.053

*Samples of the -8 mesh ore and the -100 mesh ore were each analyzed for WO_3. These results are also shown in Table III, along with the amenability tests of each sample tested.

TABLE III

Spectrographic Analyses
of Various Tasminex Ore Samples

(Supplied by Minerals Beneficiation Group)

Sought	Range in Per Cent		
	Category A Dominantly Non-Scheelite	Category B Mixed Scheelite and Non-Scheelite	Category C Dominantly Scheelite
Al	0.2-2.0	0.2-2.0	0.2-2.0
Be	0.0004-0.004	0.0008-0.008	0.001-0.01
Bi	0.02-0.2	0.008-0.08	0.02-0.2
Ca	0.008-0.08	0.01-0.1	0.08-0.8
Cu	0.03-0.3	0.02-0.2	0.02-0.2
Fe	Major	Major	Major
Mg	0.02-0.2	0.02-0.2	0.08-0.8
Mn	0.03-0.3	0.03-0.3	0.03-0.3
Mo	0.008-0.08	0.004-0.04	0.004-0.04
Ni	-	0.002-0.02	-
Pb	0.02-0.2	0.02-0.2	0.01-0.1
Si	0.2-2.0	0.2-2.0	0.3-3.0
Sn	0.04-0.4	0.04-0.4	0.04-0.4
Ti	0.02-0.2	0.01-1.0	0.008-0.08
W	0.4-4.0	0.3-3.0	0.3-3.0
Zn	0.008-0.08	0.008-0.08	0.008-0.08

TABLE IV

Screen Analyses of Samples
of Coarse Ores from Tasminex Deposit

(Supplied by Minerals Beneficiation Group)

Tyler Mesh Size	Per Cent Retained on Screen		
	Category A Dominantly Non-Scheelite	Category B Mixed Scheelite and Non-Scheelite	Category C Dominantly Scheelite
-8 + 10	1.90	1.40	3.96
-10 + 14	3.07	1.75	4.05
-14 + 20	6.95	3.29	7.79
-20 + 28	9.38	4.05	8.36
-28 + 35	10.38	6.24	9.85
-35 + 48	11.22	10.64	11.54
-48 + 65	9.60	11.68	10.59
-65 + 100	8.29	12.70	9.67
-100 + 150	6.85	11.41	7.63
-150 + 200	5.76	9.64	6.30
-200 + 270	3.10	5.00	3.32
-270 + 400	3.53	5.13	3.51
-400	19.97	17.07	13.43

TABLE V

Amenability Tests on Various Tasminex Ores Using Na_2CO_3-NaCl as Roasting Reagents to Solubilize the Tungsten Values in the Subsequent Water Leach

Constant Roast Conditions: (Tasminex ore samples as shown
(900° C. for 1 hr.
(Na_2CO_3 and NaCl as shown

Test No.	Category Designation	% WO_3 in Sample	Mesh Size	Na_2CO_3 Added: % Based on Sample Wt. or g./100 g. Ore	Na_2CO_3 Added: #/# WO_3 in Sample	Na_2CO_3 Added: #/# WO_3 Extracted	NaCl Added: % Based on Sample Wt. or g./100 g. Ore	NaCl Added: #/# WO_3 in Sample	NaCl Added: #/# WO_3 Extracted	% WO_3 Extracted	% WO_3 Content of Tails	pH of Filtrate	WO_3 Material Balance %
1	A	1.05	-8M	8.0	7.6	8.6	12.0	11.4	12.9	88.4	0.12	12.2	95
2	A	1.05	-8M	8.0	7.6	8.5	24.0	22.9	25.6	89.5	0.11	12.3	102
3	A	1.07	-100M	8.0	7.5	8.2	12.0	11.2	12.2	91.8	0.087	11.7	102
4	A	1.07	-100M	8.0	7.5	8.1	24.0	22.4	24.3	92.2	0.082	11.7	100
5	B	0.68	-8M	8.0	11.8	13.3	12.0	17.6	19.8	88.8	0.08	12.4	92
6	B	0.68	-8M	8.0	11.8	13.1	24.0	35.3	39.1	90.3	0.07	12.3	105
7	B	0.71	-100M	8.0	11.3	12.2	12.0	16.9	18.2	92.8	0.052	11.9	90
8	B	0.71	-100M	8.0	11.3	12.1	24.0	33.8	36.1	93.5	0.046	11.8	98
9	C	1.11	-8M	8.0	7.2	8.1	12.0	10.8	12.1	89.2	0.12	11.0	78
10	C	1.11	-8M	8.0	7.2	7.7	24.0	21.6	23.1	93.5	0.07	10.8	85
11	C	0.97	-100M	8.0	8.2	8.9	12.0	12.4	13.4	92.6	0.07	11.0	89
12	C	0.97	-100M	8.0	8.2	8.8	24.0	24.7	26.3	93.6	0.06	11.0	94

TABLE VI

Amenability Tests on Various Tasminex Upgraded Concentrates Produced from Tasminex Ores Using Na_2CO_3-NaCl as Reagents to Solubilize the Tungsten Values in the Subsequent Water Leach

Constant Roast Conditions: (Tasminex Concentrates as shown
(900°C. for 1 hr.
(Na_2CO_3 and NaCl as shown

Test No.	Category Designation	% WO_3 in Sample	Mesh Size	Na_2CO_3 Added: % Based on Sample Wt. or g./100 g.Conc.	Na_2CO_3 Added: #/# WO_3 in Sample	Na_2CO_3 Added: #/# WO_3 Extracted	NaCl Added: % Based on Sample Wt. or g./100 g.Conc.	NaCl Added: #/# WO_3 in Sample	NaCl Added: #/# WO_3 Extracted	% WO_3 Extracted	% WO_3 Content of Tails	pH of Filtrate	WO_3 Material Balance %
1	A	2.15	-8M	8.0	3.7	6.7	36.0	16.7	30	55.8	0.79	7.0	77°
2	A	2.15	-8M	12.0	5.6	5.9	36.0	16.7	17.8	94.0	0.14	11.1	93
3	A	2.15	-8M	16.0	7.4	7.7	36.0	16.7	17.5	95.7	0.096	11.3	111
4	A	1.88	-100M	8.0	4.3	7.2	36.0	19.1	32	60.0	0.67	7.4	94
5	A	1.88	-100M	12.0	6.4	6.8	36.0	19.1	20	93.6	0.12	11.2	103
6	B	1.22	-8M	8.0	6.6	8.5	36.0	29.5	38	77.7	0.26	10.7	92
7	B	1.22	-8M	12.0	9.8	10.5	36.0	29.5	32	93.6	0.08	11.9	105
8	B	1.17	-100M	8.0	6.8	8.4	36.0	30.8	38	80.5	0.23	10.1	104
9	B	1.17	-100M	12.0	10.3	11.1	36.0	30.8	33	92.5	0.09	12.1	107
10	C	2.79	-8M	8.0	2.9	27	36.0	12.9	119	10.8	1.71	6.6	84
11	C	2.79	-8M	12.0	4.3	4.9	36.0	12.9	14.8	87.2	0.37	10.1	?
12	C	2.73	-8M	18.0	6.5	7.9	24.0	8.6	32	82.0	0.43	10.5	?
13	C	2.63	-100M	8.0	3.0	26	36.0	13.7	118	11.6	2.06	6.8	106
14	C	2.63	-100M	12.0	4.6	7.3	36.0	13.7	21.7	63.0	0.93	7.2	93
15	C	2.63	-100M	18.0	6.8	7.3	24.0	9.1	9.8	92.8	0.19	11.0	?

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Research and Development Department
P.O. Box 579, Niagara Falls, N. Y.
July 20, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: Messrs. E. E. Anderson
W. B. DeAtley
J. S. Fox
D. J. Hansen
R & D File

FROM: N. L. Grauerholz

SUBJECT: Recovery of Tungsten Values from Tasminex Ore Samples

Summary

Tasminex core residues were received from UCEX and composited in the suggested following categories:

Category A: Oxidized surface ore - little or no visible scheelite

Category B: Oxidized ore - significant visible scheelite

Category C: Unoxidized ore

Magnetic, gravity, and flotation tests have been conducted on the three samples. The magnetic separation tests indicate that tungsten losses in the oxidized ore zones is appreciable but are reduced with finer grinding. Gravity separation tests did not produce an acceptable concentrate and losses in the table middling, sand, and slime fractions were high. Flotation results on the oxidized ore were poor. The unoxidized ore, Category C, produced a good concentrate but the tailing still contained more than 1% WO_3. Most of this loss was in nonscheelite indicating that oxidation is prevalent even in the deeper ore zones.

Samples Received

The three categories for the latest samples submitted were described as follows:

Category A - Dominantly tungstite - little or no visible scheelite

Category B - Mixed scheelite and tungstite

Category C - Dominantly scheelite

Some overlapping of samples occurred within the categories. The extent of this overlapping and the relationship of the form of tungsten occurrence and its relationship to the iron mineralization will be presented in more detail in a separate report.

The individual samples comprising each category were combined as described by Mr. R. T. Brandt's letter of May 3, 1972 to McPhar Geophysics Pty. Ltd. The samples of each category were sampled by riffle splitting out one-quarter of each sample. The remaining three-fourths portion was combined into the designated category. The bulk samples of each category were crushed to minus 8-mesh, mixed and split into 1,000-gram test charges for laboratory testing. The quarter fraction of each sample was retained as originally received for mineralogical examination.

The head sample of each category was submitted for chemical analysis for WO_3, SiO_2, Fe^{++}, Fe^{+++}, Ca, Al, Cu, Pb, Zn. A spectrographic analysis of each sample was also made. The results of these analyses are shown on Data Pages 1 and 2. These results show Category A is deficient in calcium to form $CaWO_4$. The calcium in Category B is tied up in other mineralization to some extent to give a deficient calcium content. The calcium content of Category C is sufficient to satisfy the calcium content for the tungsten values to be in the scheelite mineral. The amount of tungsten as nonscheelite in Category C is unknown.

The ferrous and ferric iron ratios show the degree of oxidation in the three samples. Category B shows more iron in the form other than magnetite than does Category A. The reason for this variation is unknown. With the assumption that all the ferrous iron is from the magnetite, the three categories contain excess ferric iron as hematite, limonite, goethite and other iron minerals as follows:

limonite is not now an accepted mineral species but still is a good field term to indicate secondary iron oxides

other includes maghemite

Ratio of $FeO:Fe_2O_3$ in Magnetite Compared to Ore

	Category A	Category B	Category C
Ore	1:6.7	1:10.7	1:4.4
Magnetite	1:2	1:2	1:2

Mineralogical studies are under way to determine the relationship between magnetite, hematite and goethite and the connection with the form of tungsten mineralization.

The screen analyses of the minus 8-mesh feed for the three categories are shown on Data Pages 3, 4, and 5. The tungsten distribution shows that a discard product cannot be made by size separation.

The losses of WO_3 in the magnetic fraction are high and a series of magnetic separation tests were run with a finer grind at 100-mesh to determine if these losses could be alleviated to some degree. The finer grind allowed the test to be conducted with the Sala Magnetic Separator without hand magnetic removal. These results are condensed as follows:

Minus 100-Mesh Head Ore Magnetic and Nonmagnetic Fraction

	% Wt.	% WO_3	WO_3 % Dist.
Category A - Magnetic Fraction	59.82	0.42	24.19
Nonmagnetic Fraction	40.18	1.96	75.81
	100.00	1.04	100.00
Category B - Magnetic Fraction	53.57	0.28	21.78
Nonmagnetic Fraction	46.43	1.16	78.22
	100.00	0.69	100.00
Category C - Magnetic Fraction	69.07	0.25	17.19
Nonmagnetic Fraction	30.93	2.69	82.81
	100.00	1.00	100.00

These results show the drop in tungsten losses in the magnetic fraction with finer grinding. The finer grind liberates more nonmagnetic material and lowers the tungsten losses in the magnetic fraction.

Flotation

Each category was subjected to flotation of the nonmagnetic fraction. A 1,000-gram test charge was ground to minus-100 mesh. The magnetic fraction was removed with the laboratory Sala Magnetic Separator and the nonmagnetic fraction was subjected to flotation.

The magnetic separation-flotation tests on each of the three categories are shown on Data Pages 9 through 14, attached. These results show that the magnetic iron fraction and rougher tailing carries substantial tungsten. The presence of scheelite was detected in these discard products but not in the amount shown by assay. The losses were predominantly as tungsten in the nonscheelite form.

Gravity Concentration

A table test was conducted on each of the three categories as shown on Data Pages 15 through 17. Minus 48-mesh head sample was subjected to a magnetic separation step to remove the high specific gravity magnetite as a separate product to make the table separation of the scheelite less difficult. The magnetic product was also tabled to determine if a substantial portion of the scheelite was free or locked with the other minerals.

The slime losses in the nonmagnetic fraction was extremely high in all three categories indicating the extremely fine nature of the scheelite present. The nonscheelite fraction in the ore also had a tendency to concentrate in the slime fraction. The scheelite content of the slimes as determined by U.V. light was much less than the tungsten content as determined by assay. The slime fraction in the treatment of the magnetic fraction also showed a high tungsten content.

Recommendations and Conclusions

The Categories A and B are not amenable to treatment by gravity concentration or flotation. Category C contains an appreciable amount of scheelite recoverable by flotation. A 28.6% WO_3 concentrate at 49.2% recovery from the total ore was obtained. The recovery based on feed to flotation was 60.3%. The losses in the rougher tailing which assayed 1.11% WO_3 was not scheelite as determined by the U.V. light. This category also contains tungsten in a form other than scheelite that is probably unrecoverable by flotation.

Additional tests to lower the tungsten losses in the magnetic fraction by finer grinding will be conducted. The degree of additional tungsten recovery by finer grinding is unknown but it is believed that some improvement can be obtained. The rejection of a low tungsten content magnetic fraction is desirable for any further concentration as the tonnage to any subsequent treatment steps will be reduced by approximately two-thirds.

A previous ore, Sample Drill Hole 102/3, produced a magnetic fraction that contained only 0.035% WO_3 when ground to 100-mesh. This rejection of magnetite was very good, when compared to our present Category C ore where the WO_3 content of the magnetic fraction was in the 0.25-0.32% WO_3 range. The losses of tungsten in the magnetic fraction vary widely. It is believed that the tungsten content of the magnetic portion of the ore body is controlled by the degree of oxidation present in that portion of the ore body. Additional tests are under way to determine if this is true. If oxidation is the controlling factor that causes tungsten losses in the magnetic fraction, a much finer grind will be necessary for liberation. In this case grinding will only be effective if the nonscheelite tungsten is associated with the goethite-hematite phases of the ore body. If some of the nonscheelite tungsten is associated with the magnetite, grinding will not be effective.

Removal of the magnetite as a product low in tungsten will accomplish considerable upgrading and make subsequent operations more economically attractive. This phase of the test program will be tested as our next step in the research program.

N. L. Grauerholz

NLG:et

Data Page 1

Tasminex Ore Samples

OR-586 Drillhole Sample Residues
Chemical Analysis

Sample	Category A	Category B	Category C
Assay			
% WO_3	1.12	0.74	1.02
SiO_2	4.48	4.03	9.29
Fe^{++}	7.82	5.19	10.56
Fe^{+++}	52.50	55.57	45.96
Ca	0.06	0.13	1.99
Al	1.71	1.68	1.67
Cu	0.068	0.054	0.063
Pb	0.056	0.037	0.013
Zn	0.067	0.075	0.053

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Data Page 2

Tasminex Ore Samples

OR-586 Drillhole Sample Residues
Spectrographic Analysis

Sample Material	Values in % on Sample Basis		
	Category A	Category B	Category C
Al	0.2 - 2.0	0.2 - 2.0	0.2 - 2.0
Be	0.0004 - 0.004	0.008 - 0.08	0.001 - 0.01
Bi	0.02 - 0.2	0.008 - 0.08	0.02 - 0.2
Ca	0.008 - 0.08	0.01 - 0.1	0.08 - 0.8
Cu	0.03 - 0.3	0.02 - 0.2	0.02 - 0.2
Fe	M	M	M
Mg	0.02 - 0.2	0.02 - 0.2	0.08 - 0.8
Mn	0.03 - 0.3	0.03 - 0.3	0.03 - 0.3
Mo	0.008 - 0.08	0.004 - 0.04	0.004 - 0.04
Ni	-	0.002 - 0.02	-
Pb	0.02 - 0.2	0.02 - 0.2	0.01 - 0.1
Si	0.2 - 2.0	0.2 - 2.0	0.3 - 3.0
Sn	0.04 - 0.4	0.04 - 0.4	0.04 - 0.4
Ti	0.02 - 0.2	0.01 - 0.1	0.008 - 0.08
W	0.4 - 4.0	0.3 - 3.0	0.3 - 3.0
Zn	0.008 - 0.08	0.008 - 0.08	0.008 -0.08

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Data Page 3

Tasminex Category A Ore Sample

Minus 8-Mesh Head Sample

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+ 8			
8 x 10	1.90		
10 x 14	3.07		
14 x 20	6.95		
20 x 28	9.38 (21.30)	0.75	15.69
28 x 35	10.38		
35 x 48	11.22		
48 x 65	9.60		
65 x 100	8.29 (39.49)	0.69	26.76
100 x 150	6.85		
150 x 200	5.76 (12.61)	0.88	10.90
200 x 270	3.10		
270 x 400	3.53 (6.63)	1.05	6.84
400 x D	19.97 (19.97)	2.03	39.81
Total	100.00	1.01 (Calc.)	100.00
		1.12 (Actual)	

Data Page 4

Tasminex Category B Ore Sample

Minus 8-Mesh Head Sample

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+ 8			
8 x 10	1.40		
10 x 14	1.75		
14 x 20	3.29		
20 x 28	4.05 (10.49)	0.62	9.08
28 x 35	6.24		
35 x 48	10.64		
48 x 65	11.68		
65 x 100	12.70 (41.26)	0.54	31.12
100 x 150	11.41		
150 x 200	9.64 (21.05)	0.52	15.29
200 x 270	5.00		
270 x 400	5.13 (10.13)	0.99	14.00
400 x D	17.07 (17.07)	1.28	30.51
Total	100.00	0.72 (Calc.)	100.00
		0.74 (Actual)	

Data Page 5

Tasminex Category C Ore Sample

Minus 8-Mesh Head Sample

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+ 8			
8 x 10	3.96		
10 x 14	4.05		
14 x 20	7.79		
20 x 28	8.36 (24.16)	0.85	20.46
28 x 35	9.85		
35 x 48	11.54		
48 x 65	10.59		
65 x 100	9.67 (41.65)	0.98	40.66
100 x 150	7.63		
150 x 200	6.30 (13.93)	0.90	12.49
200 x 270	3.32		
270 x 400	3.51 (6.83)	0.95	6.46
400 x D	13.43 (13.43)	1.49	19.93
Total	100.00	1.00 (Calc.)	100.00
		1.02 (Actual)	

Tasminex Category A

Magnetic Separation Tests

Minus 8-Mesh Feed

Product	Weight %	Assay % WO_3	Distribution % WO_3
+28 Mesh Mags	23.66	0.55	12.46
-28 Mesh Mags	47.40	0.54	24.50
-28 Mesh Midds	2.59	1.52	3.77
Total Nonmags	26.35	2.35	59.27
Feed (Head Ore)	100.00	1.04 (Calc.)	100.00

Tungsten Distribution by Size in Magnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	27.95	0.55	29.37
28 x 100	46.31	0.47	41.58
100 x 200	14.92	0.56	15.96
200 x 400	6.60	0.59	7.44
400 x D	4.22	0.70	5.65
Total	100.00	0.52	100.00

Tungsten Distribution by Size in Nonmagnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	8.53	2.08	7.53
28 x 100	27.67	1.96	23.02
100 x 200	11.21	2.43	11.57
200 x 400	7.14	2.61	7.91
400 x D	45.45	2.59	49.97
Total	100.00	2.35	100.00

Data Page 7

Tasminex Category B

Magnetic Separation Tests

Minus 8-Mesh Feed

Product	Weight %	Assay % WO_3	Distribution % WO_3
+28 Mesh Mags	13.27	0.51	9.74
-28 Mesh Mags	49.89	0.40	28.73
-28 Mesh Midds	6.62	0.75	7.15
Total Nonmags	30.22	1.25	54.38
Feed (Head Ore)	100.00	0.69 (Calc.)	100.00

Tungsten Distribution by Size in Magnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	16.83	0.51	22.64
28 x 100	47.17	0.34	42.28
100 x 200	21.63	0.34	19.39
200 x 400	10.88	0.38	10.90
400 x D	3.49	0.52	4.79
Total	100.00	0.38 (Calc.)	100.00

Tungsten Distribution by Size in Nonmagnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	4.32	1.44	4.97
28 x 100	36.12	1.07	30.89
100 x 200	20.21	1.11	17.93
200 x 400	9.96	1.32	10.51
400 x D	29.39	1.52	35.70
Total	100.00	1.25	100.00

Data Page 3

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Tasminex Category C

Magnetic Separation Tests

Minus 8-Mesh Feed

Product	Weight %	Assay % WO_3	Distribution % WO_3
+28 Mesh Mags	26.63	0.32	8.83
-28 Mesh Mags	46.93	0.27	13.14
-28 Mesh Midds	2.43	2.52	6.35
Total Nonmags	24.01	2.88	71.68
Feed (Head Ore)	100.00	0.97	100.00

Tungsten Distribution by Size in Magnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	30.05	0.32	35.04
28 x 100	49.01	0.24	42.87
200 x 400	5.28	0.30	5.77
400 x D	2.34	0.49	4.18
Total	100.00	0.27	100.00

Tungsten Distribution by Size in Nonmagnetic Fraction

Mesh Size (Tyler Series)	Weight %	Assay % WO_3	Distribution % WO_3
+28	9.09	4.50	14.21
28 x 100	32.76	3.96	45.08
100 x 200	16.46	2.65	15.16
200 x 400	11.14	2.02	7.82
400 x D	30.55	1.67	17.73
Total	100.00	2.88	100.00

Data Page 9

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category A - Composite

Sample: 1,000 Grams - 8-Mesh Ore
OR-586 1910-28

	Time Min.	pH	Reagents, Lb./Ton of Original Ore CaO	Na_2CO_3	Na_2SiO_3	NFA(1)
Grind	13	6.9	0.5			
Cond.	10	9.3	3.0			
Cond.	5	10.0		2.0	2.0	
R. Flot.	15					1.0 (staged)
Cl. Flot.	5		No Reagents			

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	67.8	0.49	32.6
R. Tail	28.0	2.01	55.2
Cl. Tail	3.2	2.65	8.4
Cl. Conc.	1.0	3.82	3.8
Calc. Feed	100.0	1.02	100.0
Actual		(1.12)	

(1) NFA: 50-50 Mixture Naphthenic Acid-Oleic Acid

Data Page 10

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category A - Composite

Sample: 1,000 Grams 8-Mesh Ore
OR-586 1910-30

	Time Min.	pH	Reagents, Lb./Ton of Original Ore CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	13	6.9	0.5			
Cond.	5	10.5	4.0			
Cond.	5			3.0	1.5	
R. Flot.	15	10.7				1.0 (staged)
Cl. Flot. No. 1	5	10.5	No Reagents			
Cl. Flot. No. 2	5	10.3	No Reagents			

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	66.3	0.46	30.2
R. Tail	25.0	1.90	47.1
Cl. Tail No. 1	4.7	2.27	10.6
Cl. Tail No. 2	1.9	2.62	5.0
Cl. Conc.	2.1	3.56	7.1
Calc.	100.0	1.01	100.0
Actual		1.12	

Data Page 11

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category C - Composite

Sample: 1,000 Grams 8-Mesh Ore
OR-586 1910-22

	Time Min.	pH	Reagents, Lb./Ton of Original Ore			
			CaO	Na_2CO_3	Na_2SiO_3	HPA
Grind	17		0.5			
Cond.	20	11.0	6.0			
Cond.	5	10.9		2.0	2.0	
R. Flot.	12	10.9				0.7
Cl. Flot. No. 1	5	10.7				0.06
Cl. Flot. No. 2	5	10.3	No Reagents			

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	70.3	0.29	18.4
R. Tail	19.0	1.11	19.0
Cl. Tail No. 1	6.7	1.49	9.0
Cl. Tail No. 2	2.1	2.29	4.4
Cl. Conc.	1.9	28.60	49.2
Calc.	100.0	1.11	100.0
Actual		1.02	

Data Page 12

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category C - Composite

Sample: 1,000 Grams 8-Mesh Ore
OR-586 1910-20

	Time Min.	pH	Reagents Lb./Ton of Original Ore CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	17		0.5			
Cond.	5	10.6	3.0	2.0	2.0	
R. Flot.	10	9.9				1.0 (staged)
Cl. Flot.	5	9.8			1.0	0.04

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	70.7	0.46	28.2
R. Tail	24.2	1.08	22.6
Cl. Tail	3.3	1.59	4.6
Cl. Conc.	1.8	28.00	44.6
Calc.	100.0	1.15	100.0
Actual		1.02	

Data Page 13

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category B - Composite

Sample: 1,000 Grams - 8-Mesh Ore
OR-586 1910-26

	Time Min.	pH	Reagents Lb./Ton of Original Ore CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	13	7.8	0.5			
Cond.	30	11.3	12.0			
Cond.	5	11.5		2.0	2.0	
R. Flot.	10					1.3 (staged)
Cl. Flot. No. 1	5		No Reagents			
Cl. Flot. No. 2	5		No Reagents			

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	57.5	0.34	28.7
R. Tail	32.5	1.02	48.8
Cl. Tail No. 1	6.7	1.45	14.3
Cl. Tail No. 2	2.3	1.62	5.4
Cl. Conc.	1.0	1.94	2.9
Calc.	100.0	0.68	100.0
Actual		0.74	

Data Page 14

Tasminex Flotation Amenability Test

Magnetic Separation - Flotation Concentration
Category B - Composite

Sample: 1,000 Grams - 8-Mesh Ore
OR-586 1910-24

	Time Min.	pH	Reagents Lb./Ton of Original Ore			
			CaO	Na_2CO_3	Na_2SiO_3	NFA
Grind	13	7.8	0.5			
Cond.	3	9.5	2.0			
Cond.	3	10.5		2.0	3.0	
R. Flot.	20	10.2				2.0 (staged)
Cl. Flot. No. 1	5	9.5	No Reagents			
Cl. Flot. No. 2	5	9.5	No Reagents			

	% Wt.	% WO_3	Dist. WO_3
Mag Conc.	61.7	0.35	31.2
R. Tail	20.8	1.15	34.6
Cl. Tail No. 1	5.5	1.13	8.9
Cl. Tail No. 2	3.7	1.40	7.5
Cl. Conc.	8.3	1.49	17.8
Calc.	100.0	0.69	100.0
Actual		0.74	

Data Page 15

Tasminex Gravity Table Test

Magnetic Separation - Gravity
Category A - Composite

Sample: 1,000 Gram Ground to Minus 48-Mesh
OR-586 1910-32

Nonmagnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	7.8	1.48	11.4
" Midds	5.1	1.69	8.6
" Sand	1.6	1.91	2.9
" Slime	20.6	2.13	43.3
Mag Fraction	64.9	0.53	33.8
Calc. Head	100.0	1.01	100.0
Actual Head		1.12	

Magnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	11.4	0.41	9.3
" Midds	84.4	0.45	75.9
" Sand	1.7	1.52	5.3
" Slime	2.5	1.90	9.5
Calc. Head	100.0	0.50	100.0
Actual Head		0.53	

Data Page 16

Tasminex Gravity Table Test

Magnetic Separation - Gravity
Category B - Composite

Sample: 1,000 Gram Ground to Minus 48-Mesh
OR-586 1910-34

Nonmagnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	9.6	0.90	12.6
" Midds	13.4	0.86	16.7
" Sand	1.0	1.30	2.0
" Slime	18.8	1.43	38.9
Mag Fraction	57.2	0.36	29.8
Calc. Head	100.0	0.69	100.0
Actual Head		0.74	

Magnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	8.3	0.36	7.5
" Midds	86.6	0.35	75.9
" Sand	2.0	1.07	5.4
" Slime	3.1	1.43	11.2
Calc. Head	100.0	0.40	100.0
Actual Head		0.36	

Data Page 17

Tasminex Gravity Table Test

Magnetic Separation - Gravity
Category C - Composite

Sample: 1,000 Gram Ground to Minus 48-Mesh
OR-586 1910-36

Nonmagnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	3.8	7.05	26.9
" Midds	5.8	0.81	4.8
" Sand	0.8	2.36	2.0
" Slime	18.1	2.48	45.3
Mag Fraction	71.5	0.29	21.0
Calc. Head	100.0	0.99	100.0
Actual Head		1.02	

Magnetic Fraction Product Distribution

	% Wt.	% WO_3	Dist. WO_3
Table Conc.	8.2	0.39	9.8
" Midds	87.8	0.28	76.0
" Sand	2.0	0.97	6.1
" Slime	2.0	1.33	8.1
Calc. Head	100.0	0.32	100.0
Actual Head		0.29	

Research and Development Department
Niagara Falls, New York
July 7, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. E. E. Anderson
W. B. DeAtley
N. L. Grauerholz
D. J. Hansen

FROM: J. S. Fox

SUBJECT: Recovery of Tungsten Values from Tasminex Tungsten Ore Samples

SUMMARY

Tests were made to improve the roast-leach process for extraction of the tungsten values from a non-magnetic portion of a Tasminex ore composite composed of surface and subsurface ores.

A combination of 8% Na_2CO_3 along with 36% NaCl* gave a tungsten solubilization of 97% in a one-hour roast at 900°C. The chemical cost of the roasting reagents was 11¢ per pound of WO_3 solubilized in the water leach.

It was also learned that 92% tungsten solubilization can be obtained by roasting with 18% Na_2CO_3 and 22% Na_2SO_4 at 900°C. Roasting without NaCl has an advantage of reduced corrosion in the roast-leach circuit; however, the chemical costs are high (approximately 24¢ per pound of WO_3 solubilized).

The tungsten values in the water leach liquor from the Na_2CO_3-NaCl roasted calcines from the non-magnetic portion of Tasminex ore composite were found to be precipitated almost completely as $CaWO_4$. This was done using H_2SO_4 for pH adjustments and either $CaCl_2$ or CaO for precipitation of the tungsten values as $CaWO_4$. The chemical costs for precipitation would be approximately 2.5 to 3.0¢ per pound of WO_3 solubilized.

DISCUSSION

Since the last memorandum on June 5, 1972, experimental work was continued on the recovery of tungsten values from the oven-dried non-magnetic portion of a tungsten composite of surface and subsurface ore. This sample was prepared by the Minerals Beneficiation Section of our laboratory under the supervision of Mr. N. L. Grauerholz.

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*Where % NaCl, % Na_2CO_3, or % Na_2SO_4 is used in this report indicates grams of NaCl, Na_2CO_3, or Na_2SO_4 per 100 grams of ore material

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Tests were made to determine if the chemical costs for roasting could be lowered by using less Na_2CO_3 in the roasts and to supplement this decrease in Na_2CO_3 by using NaCl, a cheaper reagent. Tests were also made to determine if Na_2SO_4 could be used along with Na_2CO_3 in the roasting step. It is believed that Na_2SO_4 is less corrosive on the roasting equipment and the gases emitted can be controlled more easily than when NaCl is used. Finally, tests were made to determine the best conditions for the $CaWO_4$ precipitation of the tungsten values in the water leach liquor obtained from the Na_2CO_3-NaCl roasted calcines.

I. Roasting with Na_2CO_3 and NaCl

In the previous memorandum of June 5, 1972, it was shown that 97% of the tungsten values could be solubilized from a non-magnetic portion of Tasminex ore composite of surface and subsurface ore. This non-magnetic sample contained 2.43% WO_3 and was roasted with 12% Na_2CO_3 and 36% NaCl (12 grams Na_2CO_3 and 36 grams NaCl per 100 grams of sample) at 900°C. for 1 hour. Tests were made during this interim in which the roasting temperature was increased to 950°C. or 1000°C. The conditions and results of these tests are shown in Table I (Tests 5 and 6). These results show that increasing the roasting temperature higher than 900°C. decreases the amount of tungsten solubilized in the subsequent water leach. Increasing the amount of NaCl from 24% to 36% increased the tungsten solubilization (Tests 3 and 4).

A few tests were made to determine if 8% Na_2CO_3 could be employed along with 36% NaCl while still maintaining the very high tungsten solubilization of 97%. These tests were made at 900°C., 950°C., and 1000°C., and the conditions and results of these tests are shown in Table I (Tests 7, 8, and 9). These results show that 8% Na_2CO_3 along with 36% NaCl in the roast of the sample mentioned will solubilize 97% of the tungsten values in a 1-hour roast at 900°C. At temperatures higher than 900°C., the amount of tungsten solubilization decreases. It would require 3.4 pounds of Na_2CO_3 and 14.8 pounds of NaCl per pound of WO_3 solubilized to achieve the mentioned high tungsten solubilization. Based on Na_2CO_3 and NaCl at $35.00 and $7.10 per ton, respectively, these reagents would cost approximately 11¢ per pound of WO_3 solubilized. A test using 5% Na_2CO_3 and 36% NaCl in the roast is in progress.

II. Roasting with Na_2SO_4 and Na_2CO_3

Roasting tests were made using 24% Na_2CO_3 along with 29% Na_2SO_4 on Tasminex non-magnetic concentrate at 800°C., 900°C., and 1000°C. with a 1-hour retention time at these temperatures. The conditions and results of these tests are shown in Table II. These results show that high tungsten solubilization (92-93%) can be achieved using Na_2SO_4 combined with Na_2CO_3 in the roast at temperatures of 900°C. or 1000°C. However, the reagent costs under the conditions employed would be high. To achieve 93% tungsten solubilization, 10.6 pounds of Na_2CO_3 and 12.8 pounds of Na_2SO_4 were used per pound of WO_3 solubilized (Test 2). At $35.00 per ton of Na_2CO_3 and $25.00 per ton of Na_2SO_4, the reagent cost would be approximately 35¢ per pound of WO_3 solubilized. In Test 4, using 18% Na_2CO_3 and 22% Na_2SO_4, a solubilization of 92% was obtained. It required 8.0 pounds of Na_2CO_3 and 9.9 pounds of Na_2SO_4 per pound of WO_3 solubilized. These chemical costs still would be high (approximately 24¢ per pound of WO_3 solubilized). When 12% Na_2CO_3 and 15% Na_2SO_4 were employed in the roast, inadequate tungsten solubilization was obtained.

III. Precipitation of $CaWO_4$ from Leach Liquor

A. Using $CaCl_2$

A composite of the leach liquors (pH 12.6) obtained from the water leaching of the Na_2CO_3-NaCl roasted calcines was made. The average amounts of Na_2CO_3 and NaCl employed in the roasting steps were 17.5% and 23.1%, respectively. The average tungsten solubilization obtained from the roasted calcines and dissolved in the leach liquor composite was 93%. This composite sample of leach liquor contained 2.56 grams per liter of WO_3 and was evaporated to a volume containing 10.0 g./l. WO_3. A small amount of precipitate formed which contained aluminum, silicon, and sodium and very small amounts of tungsten. This precipitate was discarded since it contained only negligible amounts of tungsten values.

A variety of tests were made in which H_2SO_4 was used to adjust the pH of the liquor prior to the addition of the $CaCl_2$ used to precipitate the $CaWO_4$. The chemical analyses of the $CaWO_4$ product have not been completed; however, the chemical analyses of the mother liquor have been completed.

The best procedure found to date using the minimum amounts of $CaCl_2$ and shortest digestion period is as follows. To 50 ml. of this composite leach liquor (10 g./l. WO_3) was added 1.3 ml. of H_2SO_4 (containing 300 g./l. H_2SO_4) to a pH of 8.9. The mixture was heated to 85°C. and then 3.6 ml. of $CaCl_2$ solution (100 g./l. $CaCl_2$) were added. The mixture was yellow and was boiled 1 hour under reflux and then filtered and washed with a few ml. of water. This yellow color is probably due to an impurity. The residue was dried and weighed and the measured filtrates were analyzed by spectrographic analyses. The mother liquor filtrate (61 ml.) contained 0.02 g./l. WO_3. The spectrographic analyses of the product and filtrate and the chemical analyses for WO_3 of the filtrate are shown in Table III. These results indicate that practically all of the tungsten was precipitated (over 99% precipitated) as a cream color precipitate from the leach liquor by the $CaCl_2$. The WO_3 content of the precipitated $CaWO_4$ is estimated to be approximately 51%. The $CaCl_2$ added was equivalent to 0.72 pound per pound of WO_3 in the composite liquor or 150% of the stoichiometric amount required to precipitate the WO_3 in solution. The H_2SO_4 used was equivalent to 0.8 pound per pound of WO_3 in solution. Based on chemical costs of $35.00 per ton for H_2SO_4 and $CaCl_2$ at $42.00 per ton, the chemical reagent costs for precipitation would be approximately 3¢ per pound of WO_3 precipitated.

B. Using CaO

The following is the best procedure found to date for using CaO to precipitate the tungsten values as $CaWO_4$ from the tungsten leach liquor composite described above.

To 50 ml. of the composite liquor containing 10 g./l. WO_3 and a pH of 12.6 was added 2.0 ml. H_2SO_4 to a pH of 5.8. The mixture was heated to 85°C. and then 0.20 gram of reagent grade CaO was added. This amount of CaO represents 165% of the stoichiometric amount of CaO required to precipitate the tungsten values in the sample to $CaWO_4$. The pH after the CaO addition was 8.8. After boiling for 1 hour under reflux, the yellow mixture was filtered and the residue was washed with a few ml. of water. The residue was dried and weighed, and the filtrate was measured. The final filtrate has a volume of 50 ml. and a pH of 11.3 and contained 0.10 g./l. WO_3. The spectrographic analyses of the

$CaWO_4$ product and the mother liquor filtrate are shown in Table III along with WO_3 chemical analyses of the filtrate. Based on the chemical analyses of WO_3 in the filtrate, 99% of the tungsten values were precipitated by the above procedure. The cream colored $CaWO_4$ product is being chemically analyzed. However, based on the WO_3 left in the filtrate it was calculated that the precipitated $CaWO_4$ should contain approximately 58% WO_3.

The chemical requirements to precipitate one pound of WO_3 from the leach liquor as $CaWO_4$ was 1.2 pounds of H_2SO_4 and 0.4 pound of CaO. The chemical costs would be about 2-1/2¢ per pound of WO_3 precipitated based on H_2SO_4 and CaO at $35.00 and $17.50 per ton, respectively.

J. S. Fox

JSF:ms

TABLE I

Effect of Na_2CO_3 Combined with NaCl in Roast of the Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of the Tungsten Values in the Water Leach

Roast Conditions: (Non-Mag. conc. (2.43% WO_3)
(Na$_2$CO$_3$ as shown
(NaCl as shown
(Temp. of Roast as shown
(Time at Temp. = 1 Hr.

	Na_2CO_3 Added			NaCl Added							
Test No.	% Based on Sample Wt. or g./100g. Conc.	#/# WO_3 in Sample	#/# WO_3 Ext.	% Based on Sample Wt. or g./100g. Conc.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C	% WO_3 Ext.	% WO_3 in Tails	pH of Filt.	WO_3 Material Balance %
1	12.0	4.9	5.6	18.0	7.4	8.5	800	87	0.35	10.8	88
2	12.0	4.9	5.6	24.0	9.9	11.3	800	88	0.31	10.6	90
3	12.0	4.9	5.4	24.0	9.9	11.0	900	90	0.23	9.9	89
4	12.0	4.9	5.1	36.0	14.8	15.3	900	97 ←	0.08	10.9	83
5	12.0	4.9	5.6	24.0	9.9	11.3	950	88	0.27	10.5	93
6	12.0	4.9	6.3	24.0	9.9	12.7	1000	78	0.49	10.6	91
7	8	3.3	3.4	36	14.8	15.3	900	97 ←	0.08	11.7	95
8	8	3.3	5.7	36	14.8	25.5	950	58	0.89	7.6	82
9	8	3.3	9.7	36	14.8	43.5	1000	34	1.24	7.5	86

TABLE II

Effect of Na_2CO_3 Combined with Na_2SO_4 in Roast of the Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization Of Tungsten Values in Water Leach

Roast Conditions: (Non-Mag. Conc. (2.43% WO_3)
(Na2CO3 as shown
(Na2SO4 as shown
(1 hr. at Temp. shown

Test No.	Na_2CO_3 Added: % Based on Sample Wt. or g./100g. Sample	Na_2CO_3 Added: #/# WO_3 in Sample	Na_2CO_3 Added: #/# WO_3 Ext.	Na_2SO_4 Added: % Based on Sample Wt. or g./100g. Sample	Na_2SO_4 Added: #/# WO_3 in Sample	Na_2SO_4 Added: #/# WO_3 Ext.	Roast Temp. °C	% WO_3 Ext.	% WO_3 in Tails	pH of Filt.
1	24.0	9.9	12.1	29.2	12.0	14.6	800	82	0.46	12.7
2	24.0	9.9	10.6	29.2	12.0	12.8	900	93	0.18	12.1
3	24.0	9.9	10.7	29.2	12.0	12.9	1000	92	0.16	11.9
4	18.0	7.4	8.0	22.0	9.1	9.9	900	92	0.19	11.7
5	12.0	4.9	7.5	15	6.2	9.5	900	65	0.85	10.4

TABLE III

Spectrographic Analyses of $CaWO_4$ and Mother Liquor After Precipitation with Either $CaCl_2$ or CaO

$CaWO_4$ Precipitated Product				Mother Liquor After Precipitation			
Precipitated With $CaCl_2$ Range in % Wt. of ppt= 0.97 grams		Precipitated With CaO Range in % Wt. of ppt= 0.86 grams		After Pptn. With $CaCl_2$ Range in g./l. Vol. of Filt.= 61 ml		After Pptn. With CaO Range in g./l. Vol. of Filt.= 50 ml	
Sought	Found	Sought	Found	Sought	Found	Sought	Found
Al	Major	Al	Major	Al	0.002-0.02	Al	0.1-1.0
B	0.002-0.02	B	0.002-0.02	B	0.0006-0.006	B	0.0006-0.006
Ca	Major	Ca	Major	Ca	0.06-0.6	Ca	0.006-0.06
Cr	0.04-0.4	Cr	0.01-0.1	Cr	0.04-0.4	Cr	0.04-0.4
Fe	-	Fe	0.004-0.04	Mo	0.006-0.06	Mo	0.04-0.4
Mg	0.002-0.02	Mg	0.004-0.04	Na	Major	Na	Major
Mo	0.2-2.0	Mo	0.2-2.0	Si	0.0006-0.006	Si	0.0006-0.006
Na	0.3-3.0	Na	0.3-3.0	V	0.0006-0.006	V	0.0006-0.006
Si	0.02-0.2	Si	0.02-0.2	W	0.01-0.1	W	0.03-0.3
V	0.02-0.2	V	0.02-0.2	WO_3* = 0.02 g./l.		WO_3* = 0.10 g./l.	
W	Major	W	Major				

*Chemical Analyses

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662091

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
ration Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date June 16, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex Project
R & D Program

opy to Messrs. E. Anderson
N. L. Grauerholz
D. J. Hansen
M. Stern
R & D File

Dear Lew:

I have attached a summary outline of our R & D program on the composite samples of the Tasminex core residues submitted by UCEX. The composites A, B, and C reportedly, were composited in the following manner:

Composite A: Oxidized surface ore - little or no visible scheelite
Composite B: Oxidized ore - significant visible scheelite
Composite C: Unoxidized ore

As we discussed during yours and Dr. Stern's visit, we will examine the alternate route of processing the unoxidized ore to a high-grade WO_3 concentrate by gravity and/or flotation. In addition, I will have John Goddard and Ed Anderson examine the chemistry of our salt-roast practice to see if we can determine what chemistry controls the roasting operation.

The composites have been prepared and we expect to start immediately on the attached program.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

662092

TASMINEX TUNGSTEN PROPERTY

R & D PROGRAM SUMMARY - COMPOSITE SAMPLES

	Composite Identification A	B	C
1. Sample Characteristics	Prepare Composite ---------->		
	Assay - WO_3, Fe, Ca, Mg, Al, SiO_2, Size Dist. ---------->		
	Mineralogy ---------->		
2. Magnetic Separation	Weight Dist. WO_3 Content and Distribution ---------->		
3. Flotation	? Goethite	?	(a) OPT. Conditions (b) Grade vs. Recovery
4. Gravity Concentration	X	X	J.G. Tables, etc.
5. Thickening Characteristics	Optimize - Flocculation, Thick. Size Requirement ---------->		
6. Salt Roast	Amenability and Chemistry ---------->		
7. Soda Ash Digestion	X	X	Amenability
8. Deposité Geology (Individual Drill Samples)			

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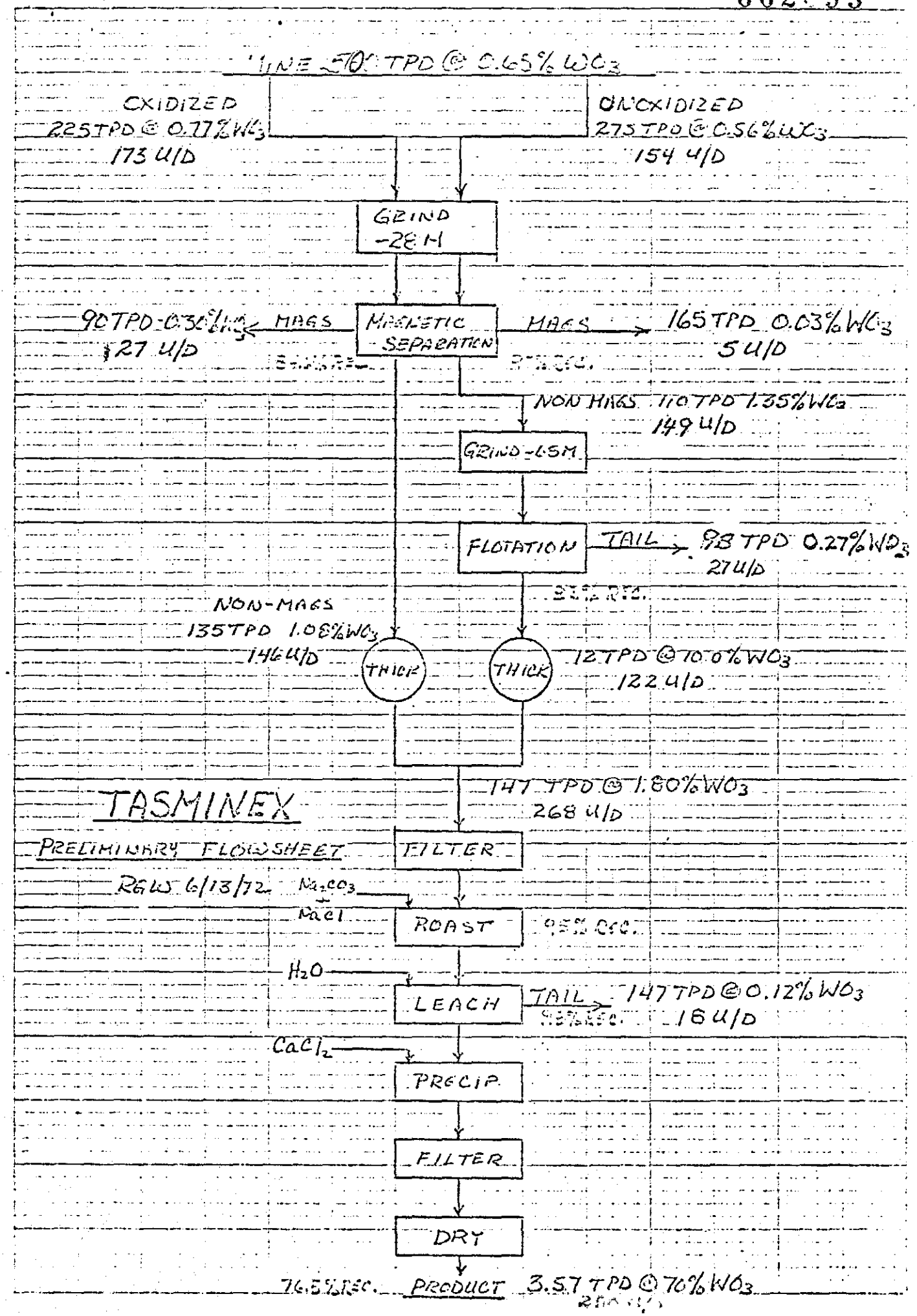
R G Woolery

662093


LINE 500 TPD @ 0.65% WO3
OXIDIZED
225 TPD @ 0.77% WO3
173 U/D
UNOXIDIZED
275 TPD @ 0.56% WO3
154 U/D
GRIND -28 M
MAGNETIC SEPARATION
MAGS
90 TPD 0.32% WO3
27 U/D
MAGS
165 TPD 0.03% WO3
5 U/D
STRIPPED
REJECT
NON MAGS 110 TPD 1.35% WO3
149 U/D
GRIND -65 M
FLOTATION
TAIL
98 TPD 0.27% WO3
27 U/D
STRIPPED
NON-MAGS
135 TPD 1.08% WO3
146 U/D
THICK
THICK
12 TPD @ 70.0% WO3
122 U/D
147 TPD @ 1.80% WO3
268 U/D
TASMINEX
PRELIMINARY FLOWSHEET
RGW 6/13/72
FILTER
Na2CO3
NaCl
ROAST
95% REC.
H2O
LEACH
TAIL
147 TPD @ 0.12% WO3
16 U/D
CaCl2
PRECIP.
FILTER
DRY
76.5% REC.
PRODUCT
3.57 TPD @ 70% WO3

TASMINEX

ORE RESERVES — 1,625,000 Tons @ 0.65% WO_3

Production = 500 TPD
Oxidized @ 225 TPD @ 0.77% WO_3
Unoxidized @ 275 TPD @ 0.56% WO_3

Total Units to Plant = Oxidized = 173 U/D
Unoxidized = 154 U/D
Total 327 U/D

UNITS PRODUCED

Oxidized = 173 x 0.84.4 x 0.95 x 0.98 = 136 U/D
(Rec) = (Mag) (Roast) (Leach)

Unoxidized = 154 x .968 x 0.82 x 0.95 x 0.98 = 114 U/D
(Rec) = (Mag) (Flot) (Roast) (Leach)

Total Units Recovered per Day = 250 U/D x 325 Days = $1,250

6,770/

GRINDING (— 8M) COSTS

500 TPD x $0.50/Ton		$250/day	$1.00/U

MAGNETIC SEPARATION COSTS

500 TPD x 0.20/Ton		$100/day	$0.40/U

GRINDING (65 M) COSTS

Unoxidized = (0.40 x 275 TPD) x 0.40	$44.00/day		
Total Grinding Costs	$44.00	$44.00	$0.18/U

FLOTATION COSTS (UNOXIDIZED ONLY)

Condition	0.40/Ton
Flot.	0.82
Other	0.70
Total	1.92

$1.92 x 110	=	$211.20/day	$0.84/U

THICKENING + FILT. COSTS

$.20/Unit x 250	50.00	0.20

Units in Flot. Conc. = 122 U/D
@ 10% Conc. = 12.2 TPD to Roast

ROASTS COSTS

Total Tons to Roast = 12.2 + 135 = 147 TPD

147 x $10.00/Ton	$1470.00/day	$5.88/U

REAGENT COSTS

5 lb. Na_2CO_3 + 15 lb. NaCl per lb. WO_3
100 lb. " + 300 lb. " per unit WO_3

Na_2CO_3 @ $1.75/100#	= $1.75 x 250	$437.50/day	$1.75/U	
NaCl @ $0.40/100#	= 1.20 x 250	$300.00/day	1.20/U	
Total Reagent Cost			$737.50/day	$2.95/U

PRECIP. + FILTER COSTS

250 Units @ $1.00/U	=	$250.00/day	$1.00/U

DRY + PACK

250 Units @ $0.30/U	=	$ 75.00/day	$0.30/U
TOTAL PRODUCTION COST		$3187.70	$12.75

MINING COSTS

1.80/Ton x 1000 Tons ÷ 250 U/D	=	$1800.00/day	$7.20/U
			$19.95

INVESTMENT WRITE-OFF

Mine	$ 250,000
Mill (Roast)	2,025,000
Mill (Flot.)	550,000
Services	800,000
827	350,000
Total	$3,975,000
Add Inv. Etc.	500,000 } working cap.
Cash	200,000
Total	$4,675,000 TUI

Write-Off = $\frac{4,675,000}{10 \times 81,250 \text{ u/yr}}$ = $5.75/Unit — $1052.50/day — $5.75/U

ROYALTY = $1.50/Unit — $ 375.00 — 1.50

Taxes $1.00/Unit — $ 250.00 — 1.00

Total Mining & Investment Costs	$3477.50	$15.45
Total Production Costs	$3187.70	$12.75
GRAND TOTAL	$6665.20	$28.20

$$\% \text{ ROI} = \frac{(37.50 - 28.20)\ 250 \times 325}{4,675,000}$$

% ROI = 16.2%

6/15/72

TASMINEX

COST COMPARISON OXIDIZED VS. UNOXIDIZED

	Oxidized 136 U/D		Unoxidized 114 U/D		Total 250 U/D	
	$/Unit	$/Day	$/Unit	$/Day	$/Unit	$/Day
Mining	7.20	979.20	7.20	820.80	7.20	1800.00
Grind - 28M #	1.00	136.00	1.00	114.00	1.00	250.00
Mag. Separation	0.40	54.40	0.40	45.60	0.40	100.00
Grind - 65M			0.39	44.00	0.18	44.00
Flotation			1.85	211.20	0.84	211.20
Thick. + Filter	0.29	40.00	0.09	10.00	0.20	50.00
Roast	9.93	1350.00	1.05	120.00	5.88	1470.00
Roast Reagents	2.95	401.20	2.95	336.30	2.95	737.50
Leach + Precip.	1.00	136.00	1.00	114.00	1.00	250.00
Filter + Dry	0.30	40.80	0.30	34.20	0.30	75.00
Total	23.07	3137.60	16.23	1850.10	19.95	4987.70

TASMINEX

CASE II - ASSUME 275 U/D RECOVERY (84%)

ORDER OF MAGNETUDE COST ESTIMATE

	$/Day	$/Unit
Mining		
$1.80/ton - 1:1 Ore:Waste Ratio - 1000 TPD	1800	6.55
Grinding - (-8M)		
500 TPD x $0.50/ton	250	0.91
Magnetic Separation		
500 TPD x $0.20/ton	100	0.36
Grinding - (-65M)		
$0.40/ton x 275 TPD x 0.40	44	0.16
Flotation		
$1.92/ton x 110 tons	211	0.77
Thickening - $0.20/unit	55	0.20
Roast @ $10.00/ton x 147 TPD	1470	5.35
Roast Reagents @ $2.95/unit	811	2.95
Precip., Filter, Dry, etc @ $1.30/unit	358	1.30
Total Mining and Processing	5099	18.55
Write Off = $\frac{4,675,000}{10 \times 275 \times 325}$ =	1438	5.23
Royalty	412	1.50
Taxes	275	1.00
Total Man. Costs	7225	26.28

% ROI = $\frac{(37.50-26.28)(275)(325)}{4,675,000}$ = 21.5%

145 2

INTERNAL CORRESPONDENCE 662098

MINING AND METALS DIVISION • P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

o (Name) Mr. L. P. Twichell
vision Mining & Metals Division
ocation Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date June 6, 1972

Originating Dept. Research and Development Department

Answering letter date

opy to Messrs. A. W. Heuck (2)
J. C. Stephenson
M. Stern

Subject Tasminex Ore Processing

Dear Lew:

Joe Fox is continuing his optimization studies on the salt roasting of the non-magnetic portion of the Tasminex ore samples. His report for May, 1972, is attached.

His results show that a combination of sodium carbonate and sodium chloride at 850-900° will result in about ± 95% tungsten solubilization. The temperature of the water leach and the use of wet compacts were investigated and found to have little or no effect on tungsten recovery. Roasting at temperature for longer than 15 minutes did not significantly improve the tungsten extraction. Reagent costs have been estimated at $0.13 per pound of WO_3 extracted.

Ed Anderson is examining the roasted products mineralogically and his studies should have something to report in a week or two.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

RECEIVED
U. C. EXPLORATION CORP.
JUN 7 1972

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662099

Research and Development Department
Mining and Metals Division - UCC
Niagara Falls, New York
June 5, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. W. B. DeAtley
N. L. Grauerholz
D. J. Hansen

FROM: J. S. Fox

SUBJECT: Recovery of Tungsten Values from Tasminex Tungsten Ore Samples

SUMMARY

Tests were made to improve the roast-leach process for extraction of the tungsten values from a non-magnetic portion of a Tasminex ore composite composed of surface and subsurface ores.

1. Effect of Roasting with Na_2CO_3, NaCl, or Na_2SO_4 Alone or in Various Combinations on Non-Magnetic Portion of Tasminex Ore Composite on Tungsten Solubilization

Very poor extractions were obtained from roasts of the non-magnetic portion of Tasminex ore composite (2.43% WO_3) when either NaCl or Na_2SO_4 was employed as the roasting reagent. In no case was more than 6% of the tungsten values solubilized.

When only Na_2CO_3 was employed as a roasting reagent, a maximum of 85% of the tungsten values was solubilized in a 1-hour roast using 24% Na_2CO_3*. Increasing the roasting temperature to 900°C. or 1000°C. lowered the tungsten solubilization. The chemical cost of the Na_2CO_3 was 20¢ per pound of WO_3 extracted.

A combination of 12% Na_2CO_3 along with 36% NaCl* gave a tungsten solubilization of 97% in a 1-hour roast at 900°C. The chemical cost for roasting was 13¢ per pound of WO_3 extracted.

A roast using 24% Na_2CO_3 and 29% Na_2SO_4 gave 82% tungsten solubilization in a 1-hour roast at 800°C. Higher roasting temperature tests are in progress.

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*Where % NaCl, % Na_2CO_3, or % Na_2SO_4 is used in this report indicates grams of NaCl, Na_2CO_3, or Na_2SO_4 per 100 grams of ore material

2. Effect of Roasting Temperature on Roasts Using Na_2CO_3 Along with NaCl as a Roasting Reagent

In 1-hour roasts using 18% Na_2CO_3 and 18% NaCl, it was found that roasting temperatures of 850°C. to 900°C. gave slightly higher tungsten solubilization than comparative roasts made at 750°C. to 800°C. Using 24% Na_2CO_3 and 24% NaCl in the roasts, it was again found that the higher roasting temperature roasts produced slightly higher tungsten solubilizations.

3. Effect of Using 1" Wet Compacts in Roast Instead of Powdered Material

When roasting the non-magnetic concentrate (2.43% WO_3) along with 24% Na_2CO_3 and 24% NaCl in the powdered form for 1 hour at 800°C., 96% tungsten was solubilized (tails 0.12% WO_3). When the 1" wet compact was roasted under comparable conditions, essentially the same tungsten extraction (95%) was obtained (tails 0.12% WO_3).

4. Effect of Roasting Time on Tasminex Tungsten Ore Composite (1.21% WO_3)

It was again verified that the roasting time of Tasminex ore composite roasted with 12.5% Na_2CO_3 and 12.5% NaCl was not critical. In a 15-minute roast, 94% of the tungsten values from a Tasminex ore composite (1.21% WO_3) was solubilized in a roast at 800°C.

5. Effect of Leaching Temperature on Tungsten Solubilization of Na_2CO_3-NaCl Roasted Calcium

Leaching tests showed that the temperature of the leaching of calcines from the non-magnetic portion of Tasminex ore composite roasted with 24% Na_2CO_3 and 24% NaCl at 800°C. for 1 hour was not critical. At room temperature (23°C.), 94% of the tungsten values was solubilized (tails 0.14% WO_3) in 1 hour compared to 95% solubilization at 60°C. (tails 0.12% WO_3) and 96% at the boiling point (tails 0.11% WO_3).

DISCUSSION

Since the last memorandum on May 2, 1972, experimental work was continued on the recovery of tungsten values from the oven-dried non-magnetic portion of a tungsten composite of surface and subsurface ore. This sample was prepared by Mr. N. L. Grauerholz of the Mineral Beneficiation Section of our laboratory. The chemical and spectrographic analyses of this non-magnetic concentrate are shown in Table I. The screen analyses are shown in Table II.

A study was made on (1) the effect of roasting with either Na_2CO_3, NaCl, or Na_2SO_4 alone or in combinations on the subsequent water solubilization of the tungsten values from the non-magnetic portion of the Tasminex composite, (2) the effect of the roasting temperature on the subsequent water solubilizations of the

tungsten values of the non-magnetic portion of the Tasminex composite roasted with Na_2CO_3 combined with NaCl, and (3) the effect of roasting a 1" wet compact on the subsequent tungsten solubilization using Na_2CO_3 combined with NaCl as a roasting reagent for the concentrate. A test was made (4) to determine the effect of a 15-minute roast at 800°C. on the subsequent water solubilization of the tungsten values from a composite sample of Tasminex ore containing surface and subsurface ore containing 1.21% WO_3. Finally, (5) a few tests were made to determine the effect of the leaching temperature on the amount of tungsten values solubilized from the roasted calcines.

1. Effect of Either Na_2CO_3, NaCl, or Na_2SO_4 Alone or in Various Combinations as a Roasting Reagent for Non-Magnetic Portion of Tasminex Ore Composite

A. Na_2CO_3 Combined with NaCl as Roasting Reagent

A series of roast tests were made in which 12% Na_2CO_3 was added to the non-magnetic portion of the total composite of Tasminex ore along with varying amounts of NaCl ranging from 0% to 36%. The roasts were performed in an Inconel X dish with a residence time of 1 hour at 800°C. (except where noted in Table III). The calcines were air cooled and were found to be soft. These calcines were pulverized and leached with water at the boiling point at approximately 30% pulp density for 1 hour and then filtered and water washed. The oven-dried residues were weighed and the filtrates were measured. The residues and filtrates were analyzed for WO_3 by wet chemical methods. Another series of tests were made as described using 18% Na_2CO_3 and 0% to 36% NaCl. Finally, a series of tests were made using 24% Na_2CO_3 and 0% to 36% NaCl. The conditions and results of these tests are shown in Table III and Figure 1. These results show that the addition of NaCl to Na_2CO_3 increased the tungsten solubilization at each level of Na_2CO_3 employed in the roast. High tungsten solubilization of 91% was obtained with 12% Na_2CO_3 and 36% NaCl. Increasing the Na_2CO_3 content to 18% and using 36% NaCl raised the tungsten solubilization to 96%. Further increases in Na_2CO_3 to 24% along with 36% NaCl improved the WO_3 only slightly to 97%. The chemical cost of the reagents using 18% Na_2CO_3 and 36% NaCl in the roast to achieve 96% WO_3 extraction was 19¢ per pound of WO_3 extracted in the water leach.

B. Na_2CO_3 Combined with Na_2SO_4 as Roasting Reagents

A series of tests were made using 24% Na_2CO_3 combined with 29% Na_2SO_4 as the roasting reagents. These tests were performed at various temperatures ranging from 800°C. to 1000°C. The conditions and results of these tests are shown in Table IV. These tests show that only 82% of the tungsten values was solubilized at 800°C. and is much lower than the 96% WO_3 extraction when NaCl was used instead of Na_2SO_4. Roasting at higher temperatures of 900°C. and 1000°C. has been made and the samples are being analyzed.

C. Na_2CO_3 Alone as a Roasting Reagent

A series of roast tests were made using Na_2CO_3 as the roasting reagents in amounts ranging from 12% to 24%. These tests were conducted at 800°C. The conditions and results of these tests are shown in Table V and Figure 2. From 82% to 85% of the tungsten values was solubilized when 18 to 24% Na_2CO_3 was employed in the roasts. Increasing the roasting temperature to 900°C. failed to significantly increase the tungsten solubilization in the roast using 18% Na_2CO_3. Increasing the roasting temperature to 1000°C. lowered the tungsten solubilization. These tests show that at the roasting temperatures investigated, the tungsten solubilization is much lower than would be obtained if Na_2CO_3 were employed in smaller amounts along with NaCl. With 24% Na_2CO_3 in the roast, the chemical cost for roasting would be 20¢ per pound of WO_3 extracted.

D. Na_2SO_4 Alone as a Roasting Reagent

A few tests were made to determine if Na_2SO_4 alone could be employed as a roasting reagent instead of Na_2CO_3. Tests were made at 800°C. using 31 to 61% Na_2SO_4 in the 1-hour roasts. The conditions and results of these tests are shown in Table VI. These results show that practically none of the tungsten values were solubilized. Increasing the roasting temperature to 900°C. or 1000°C. failed to improve the amount of tungsten solubilized. These tests show that Na_2SO_4 alone is a very poor reagent for solubilizing the tungsten values from the non-magnetic portion of the Tasminex ore composite.

E. NaCl Alone as a Roasting Reagent

Previously NaCl was employed as a roasting reagent for the total composite Tasminex sample containing a mixture of surface and subsurface ore. As reported in the memorandum of May 2, 1972 to Mr. R. G. Woolery, poor tungsten solubilization was obtained. During this interim, a few tests were made on the non-magnetic portion of this total composite using NaCl alone as the roasting reagent. The conditions and results of these tests are shown in Table VII. These tests show that very poor tungsten extractions (less than 7% WO_3) were obtained.

2. Effect of Roasting Temperature Using Na_2CO_3 Combined with NaCl as a Roasting Reagent

A series of tests were made to determine the effect of the temperature of the roast on the subsequent solubilization of the tungsten values in the water leach. In these 1-hour roasts, the temperature ranged from 750°C. to 900°C. To each roast mix was added 18% Na_2CO_3 along with 18% NaCl. The conditions and results of these tests are shown in Table VIII and Figure 3. These tests show that excellent tungsten solubilization (96%) was obtained at roast temperatures of 850°C. or 900°C. At temperatures of 750°C. or 800°C., the extractions were lower (89% and 93%, respectively). Another series of tests were made as described using 24% Na_2CO_3 and 24% NaCl in the roast. The conditions and results of these tests are also shown in Table VIII and Figure 3. These results also show that at roasting temperatures of 850 or 900°C. higher tungsten solubilization was obtained (97% and 98%, respectively) than at the lower temperatures of 750 or 800°C. (94% and 96%, respectively).

These tests show that extremely good solubilization of the tungsten values can be obtained in 900°C. roasts using 18% Na_2CO_3 and 18% NaCl.

Tests were now made using a lower amount of Na_2CO_3 of 12% combined with NaCl ranging from 24 to 36%. The conditions and results of these tests are shown in Table III (Tests 5 and 6). The results of these tests show that by roasting at 900°C. for 1 hour using 12% Na_2CO_3 combined with 36% NaCl, an extremely high tungsten solubilization of 97% was obtained. The chemical cost of the reagents to give this high tungsten solubilization of 97% was calculated to be 13¢ per pound of WO_3 extracted. This calculation is based on Na_2CO_3 costs at $35.00 per ton and NaCl at $7.10 per ton. These tests show that a major improvement in tungsten solubilization and lower chemical costs for roasting were realized by using a combination of Na_2CO_3 and NaCl instead of Na_2CO_3 alone.

3. Effect of Forming 1" Wet Compact Prior to Roasting Instead of Roasting Powdered Ore Concentrate

A test was made to determine if a 1" wet compact could be substituted for the powder in a roast. Since extrudates are successfully being used in the NaCl roast-water leach process of vanadium ores at Arkansas, it would be desirable to know if extrudates could be employed in the roast of Tasminex materials for tungsten solubilization. A 1" wet compact was prepared by adding H_2O to a mix containing 24 g. Na_2CO_3 and 24 g. NaCl per 100 g. of non-magnetic portion of Tasminex total composite. The amount of water added was 13% based on the weight of the mix containing the reagents of Na_2CO_3 and NaCl or 13 g. of water per 100 g. of mix. After roasting at 800°C. for 1 hour, air cooling, pulverizing, and water leaching, 95% of the tungsten values was solubilized. The tails contained 0.12% WO_3. By comparison, 96% WO_3 solubilization was obtained (tails 0.10% WO_3) when a powder was employed in the roast without H_2O. Thus, a 1" wet compact can be employed instead of a powder in the roast of the non-magnetic portion of the Tasminex sample. If a concentrate is used that would contain larger amounts of materials or compounds in the reduced valence state, another roast test would be suggested using the 1" wet compact.

4. Effect of Time of Roast on Tungsten Solubilization in Subsequent Water Leach

In the previous memorandum of May 2, 1972 to Mr. R. G. Woolery, it was stated that 95% of the tungsten values (tails 0.06% WO_3) was solubilized after roasting a composite sample of surface and subsurface Tasminex ore with 12.5% Na_2CO_3 containing 12.5% NaCl for 30 minutes at 800°C. It was found during this interim that a comparative roast of only 15 minutes at 800°C. would solubilize 94% of the tungsten values (0.08% WO_3 in tails). This test confirms the results of the earlier tests which showed that extensive roasting periods are not required to produce excellent tungsten solubilizations.

5. Effect of Leaching Temperature on the Amount of Tungsten Values Solubilized in Water Leach of Roasted Calcines

A few tests were made to determine the effect of the temperature of leaching of calcines from a Na_2CO_3-NaCl roast on the amount of tungsten values dissolved. The calcines produced were made by roasting the non-magnetic portion of the Tasminex ore composite with 24% Na_2CO_3 along with 24% NaCl for 1 hour at 800°C. The pulverized calcines were leached for 1 hour at room temperature (23°C.), 60°C., and at the boiling point (102°C.) and then filtered and washed. The conditions and results obtained are shown in Table IX. These results show that leaching at room temperature produced a slightly lower tungsten recovery than when leached at the boiling point (94% at 23°C. versus 96% at the boiling point). These results show that the tungsten values are readily soluble and that the leaching temperature is not critical.

J. S. Fox

JSF:ms

TABLE I

Chemical and Spectrographic Analyses of Non-Magnetic Portion of Tasminex Ore Composite

(Prepared by N. L. Grauerholz)

Chemical Analysis

Sought	Found in %
WO_3	2.43

Spectrographic Analyses	Range in %
Ag	0.0008-0.008
Al	0.8-8
B	0.004-0.04
Ba	0.002-0.02
Be	0.001-0.01
Bi	0.02-0.2
Ca	0.3-3.0
Cr	0.002-0.02
Cu	0.08-0.8
Fe	Major
Ga	0.003-0.03
Mg	0.1-1.0
Mn	0.08-0.8
Mo	0.008-0.08
Na	0.1-1.0
Ni	0.001-0.01
Pb	0.08-0.8
Si	0.3-3.0
Sn	0.08-0.8
Ti	0.03-0.3
V	0.0008-0.008
W	1-10
Zn	0.02-0.2

TABLE II

Screen Analyses of Non-Magnetic Portion of Tasminex Ore Composite

(Prepared by N. L. Grauerholz)

Wet Screen Analyses (Tyler Screen Size)	% Retained
-65 + 100	0.13
-100 + 150	0.77
-150 + 200	6.86
-200 + 270	7.53
-270 + 400	8.16
-400	76.55

TABLE III

Effect of Na_2CO_3 Combined with NaCl in Roast of the Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of the Tungsten Values in the Water Leach

Roast Conditions: (Non-mag. conc. (2.43% WO_3)
(Na_2CO_3 as shown
(NaCl as shown
(1 hr. at 800°C.

Test	Na_2CO_3 Added			NaCl Added						
	%** Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	%** Based on Sample	#/# WO_3 in Sample	#/# WO_3 Ext.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	% WO_3 Material Balance
1	12.0	4.9	6.9	0.0	0.0	0.0	71	0.75	11.1	80
2	12.0	4.9	5.6	18.0	7.4	8.5	87	0.35	10.8	88
3	12.0	4.9	5.6	24.0	9.9	11.3	88	0.31	10.6	90
4	12.0	4.9	5.4	36.0	14.8	16.3	91	0.23	10.8	91
5	12.0*	4.9	5.4	24.0*	9.9	11.0	90	0.23	9.9	89
6	12.0*	4.9	5.1	36.0*	14.8	15.3	97	0.08	10.9	83
7	18.0	7.4	9.0	0.0	0.0	0.0	82	0.48	12.2	79
8	18.0	7.4	8.1	12.0	4.9	5.4	91	0.22	11.9	91
9	18.0	7.4	8.0	18.0	7.4	8.0	93	0.17	11.9	94
10	18.0	7.4	7.8	24.0	9.9	10.4	95	0.13	11.9	90
11	18.0	7.4	7.7	36.0	14.8	15.4	96	0.10	11.7	88
12	24.0	9.9	11.6	0	0	0	85	0.38	12.6	85
[illegible]	24.0	9.9	10.5	12.0	4.9	5.3	94	0.14	12.6	93
14	24.0	9.9	10.3	18.0	7.4	7.7	96	0.10	12.5	95
15	24.0	9.9	10.2	24.0	9.9	10.2	96	0.10	12.4	97
16	24.0	9.9	10.2	36.0	14.8	15.3	97	0.08	12.5	86

*Roasted at 900°C.

**Indicates grams per 100 grams of conc.

TABLE IV

Effect of Na_2CO_3 Combined with Na_2SO_4 in Roast of the Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Non-mag. conc. (2.43% WO_3)
(Na2CO3 as shown
(Na2SO4 as shown
(1 hr. at temp. shown

	Na_2CO_3 Added			Na_2SO_4 Added							
Test No.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	WO_3 Materi Balanc %
1	24.0	9.9	12.1	29.2	12.0	14.6	800	82	0.46	12.7	95
2	24.0	9.9		29.2	12.0		900			12.1	
3	24.0	9.9		29.2	12.0		1000			11.9	

TABLE V

Effect of Na_2CO_3 without Other Reagents in Roast of Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Non-mag. conc. (2.43% WO_3)
(Na_2CO_3 added as shown
(Roasting temp. as shown
(1 hr. at roasting temp.

	Na_2CO_3 Added							
Test No.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	% WO_3 Material Balance
1	12	4.9	6.9	800	71	0.75	11.1	80
2	18	7.4	9.0	800	82	0.48	12.2	79
3	24	9.9	11.6	800	85	0.38	12.6	85
4	12	4.9	8.8	900	56	1.13	10.6	85
5	18	7.4	8.9	900	83	0.44	11.6	92
6	12	4.9	-	1000	2	2.34	10.6	103
7	18	7.4	-	1000	25	1.89	11.3	113

*Indicates grams per 100 grams of conc.

TABLE VI

Effect of Na_2SO_4 Alone Without Other Reagents in Roast of Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Non-mag. Conc. (2.43% WO_3)
(Na2SO4 as shown
(Roasting temp. as shown
(1 hr. at roasting temp.

	Na_2SO_4 Added							
Test No.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C.	% WO_3* Ext.	WO_3 in Tails %	pH of Filt.	WO_3 Material Balance %
1	30.6	12.6	-	800	1	2.71	6.3	101
2	61.2	25.2	-	800	3	2.63	6.4	100
3	30.6	25.2	-	900	1		6.8	
4	30.6	25.2	-	1000	2		7.2	

*Based on filtrate and head anal.

TABLE VII

Effect of NaCl Alone Without Other Reagents in Roast of Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Non-mag. conc. (2.43% WO_3)
(NaCl as shown
(Roasting temp. as shown
(1 hr. at roasting temp.

	NaCl Added							
Test No.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	WO_3 Material Balance %
1	33.2	13.7	-	900	1	-	7.9	-
2	33.2	13.7	-	1000	6	-	9.6	-

*Indicates grams per 100 grams of conc.

TABLE VIII

Effect of Roasting Temperature Using Na_2CO_3 Combined with NaCl in Roast of Non-Magnetic Portion of Tasminex Ore Composite on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Non-mag. conc. (2.43% WO_3)
(Na2CO3 as shown
(NaCl as shown
(Temp. as shown
(1 hr. at roast temperature

	Na2CO3 Added			NaCl Added							
Test No.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	%* Based on Sample wt.	#/# WO_3 in Sample	#/# WO_3 Ext.	Roast Temp. °C.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	W Mat ria Bal anc %
1	18.0	7.4	8.3	18.0	7.4	8.3	750	89	0.27	12.1	86
2	18.0	7.4	8.0	18.0	7.4	8.0	800	93	0.17	11.9	94
3	18.0	7.4	7.7	18.0	7.4	7.7	850	96	0.10	11.7	90
4	18.0	7.4	7.7	18.0	7.4	7.7	900	96	0.09	11.7	87
5	24.0	9.9	10.5	24.0	9.9	10.5	750	94	0.16	12.5	92
6	24.0	9.9	10.2	24.0	9.9	10.2	800	96	0.10	12.4	97
7	24.0	9.9	10.2	24.0	9.9	10.2	850	97	0.07	12.4	89
8	24.0	9.9	10.1	24.0	9.9	10.1	900	98	0.06	12.2	85

*Indicates grams per 100 grams of conc.

TABLE IX

Effect of Water Leaching Temperature on Amount of Tungsten Values Solubilized from Calcines from Na_2CO_3-NaCl Roast of Non-Magnetic Portion of Tasminex Ore Composite

Roast Conditions:
(Non-mag conc. (2.43% WO_3)
(24% Na_2CO_3
(24% NaCl
(800°C. - 1 hr.

Water Leach Conditions:
(1 hr. at temp. shown
(Pulp density ~ 30%
(or 3.2 ml. H_2O/g. calcine
(Temp. of leach as shown

Test No.	Leach Temp. °C.	% WO_3 Ext.	% WO_3 in Tails	pH of Filt.	WO_3 Material Balance %
1	23	94	0.14	11.5	88
2	60	95	0.12	12.3	92
3	B.P. (~102°C.)	96	0.11	12.5	94

FIGURE 1

Effect of Na_2CO_3 with Various Amounts of NaCl in Roast of Non-Magnetic Portion of Total Composite on WO_3 Solubilization in Subsequent Water Leach (800°C. - 1 hr. Roast)

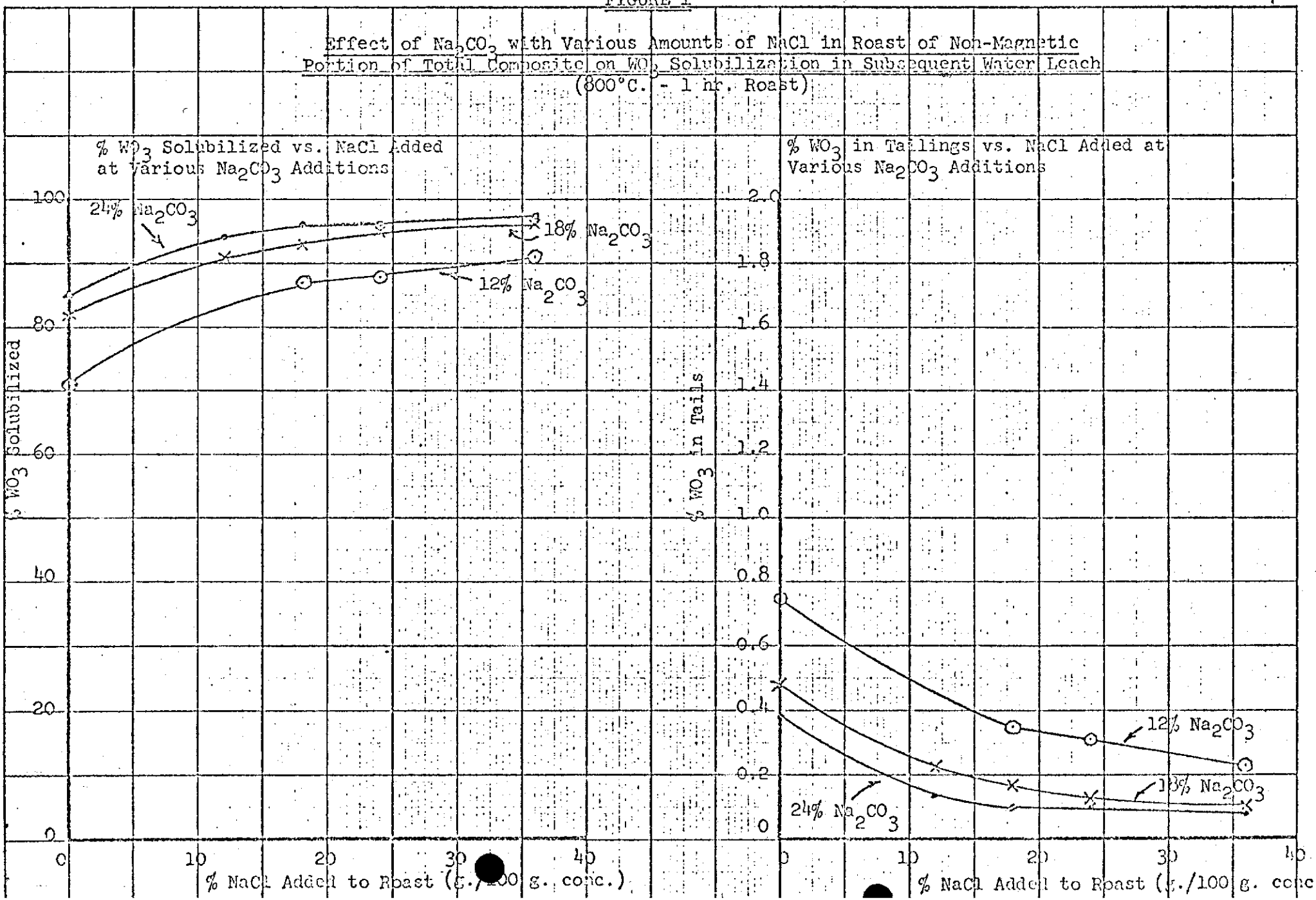


FIGURE 2

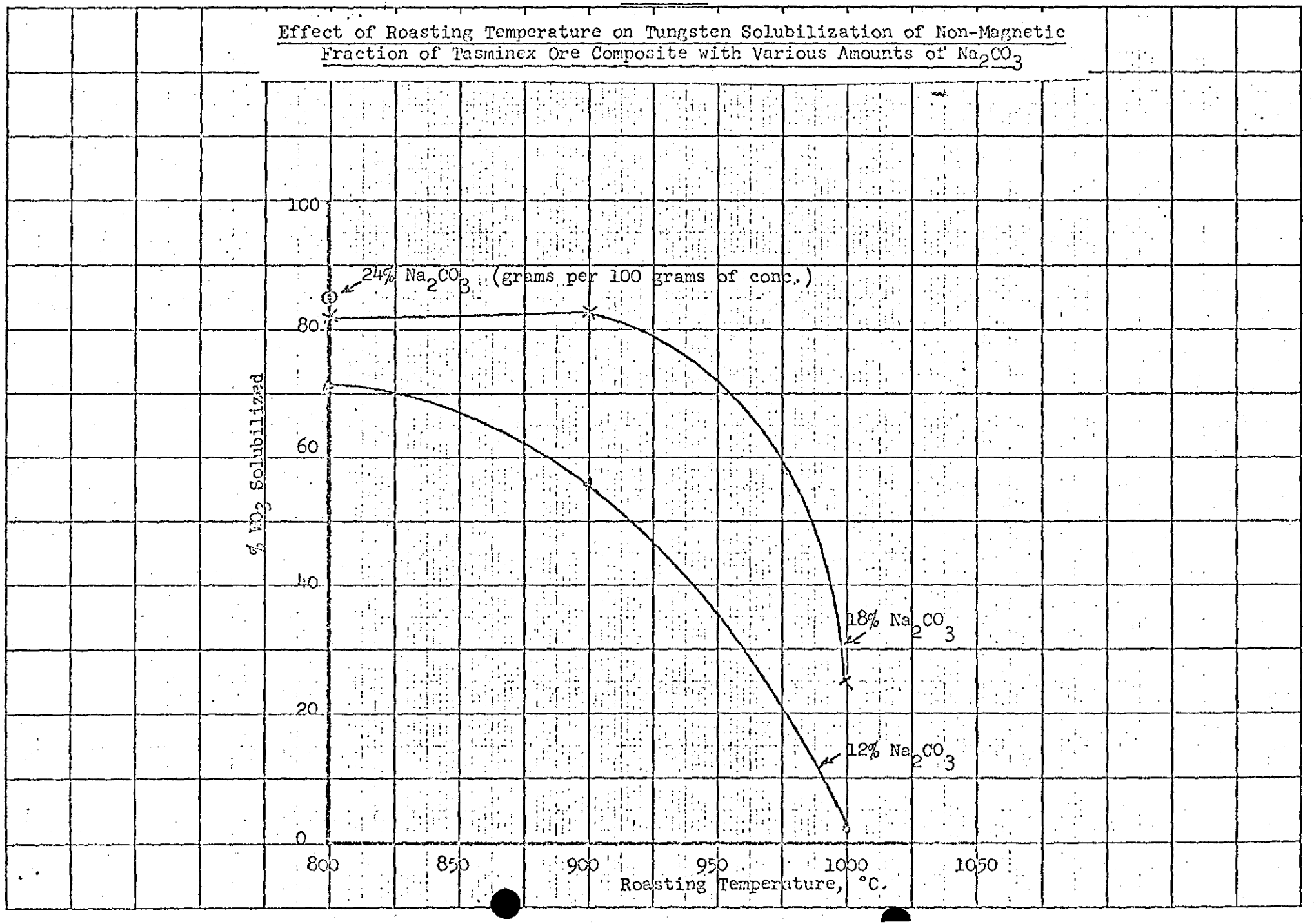
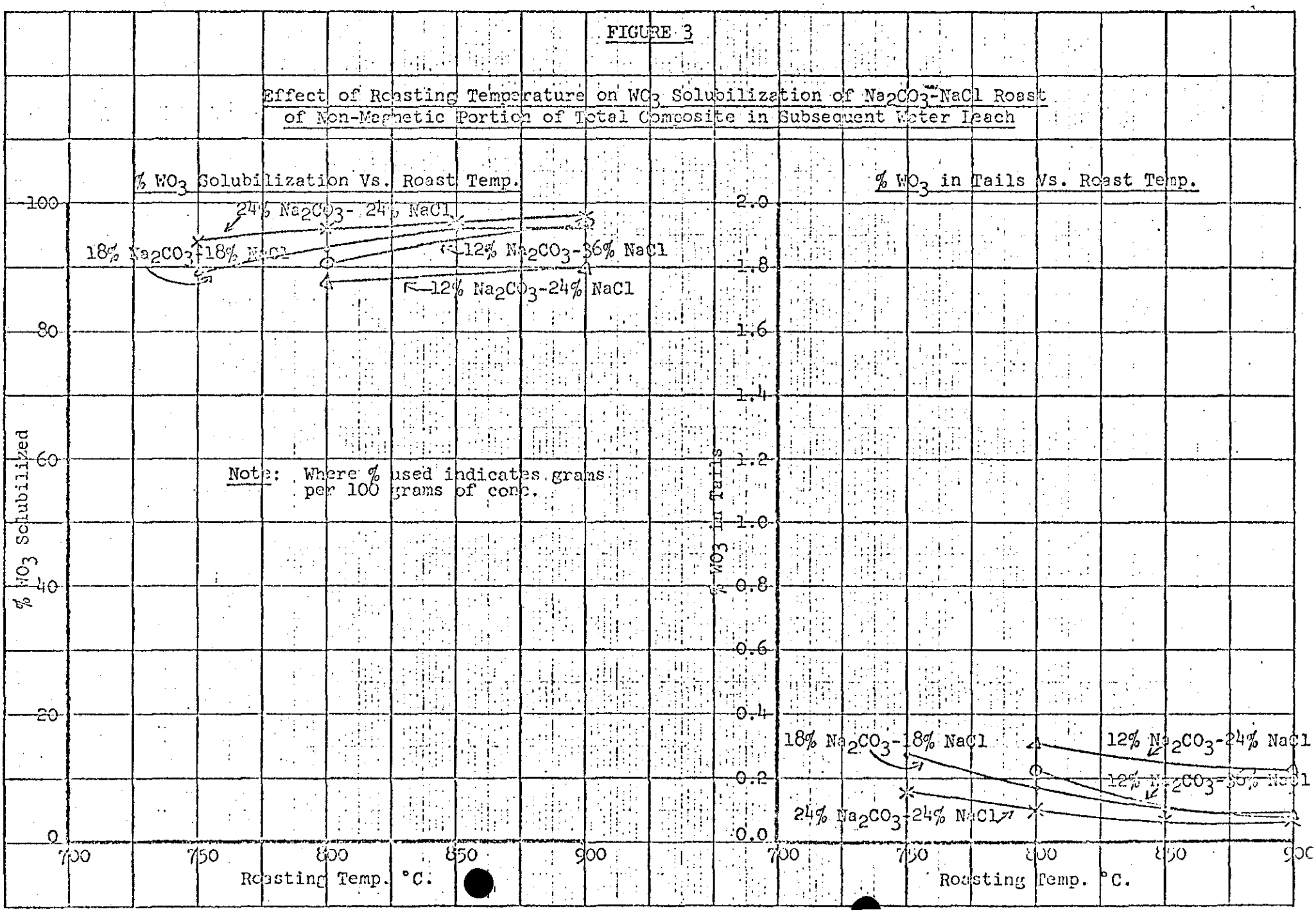


FIGURE 3

Effect of Roasting Temperature on WO_3 Solubilization of Na_2CO_3-NaCl Roast of Non-Magnetic Portion of Total Comcosite in Subsequent Water Leach



UCC - Mining & Metals

INTERNAL CORRESPONDENCE 662115

MINING AND METALS DIVISION • P. O. BOX 97, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. R.J. Klotzbach
Division UCC - Mining & Metals
Location Niagara Falls, New York

Date May 11, 1972

Originating Dept. Engineering Department

Answering letter date

Copy to Messrs. J.C. Stephenson
M. Stern
L.P. Twichell
R.G. Woolery
O.J. Malacarne

Subject Tasminex Tungsten
Newport News Rotary Kiln

SUMMARY AND CONCLUSIONS

The roaster feed grade was estimated to be 1.2 percent WO_3 by assuming the simultaneous processing of the oxidized and the upgraded unoxidized ore. For a production rate of 200,000 units of WO_3 per year, preliminary sizing indicates that a 12 foot diameter by 200 foot long rotary kiln will be required to process the specified 25 tons per hour. Since the Newport News rotary kiln will handle an estimated 11 tons per hour, it is too small for this application.

INTRODUCTION

Preliminary tests results indicate that all of the Kara I ore cannot be upgraded. The present processing scheme includes upgrading part of the ore body and recovering the tungsten by a Na_2CO_3 - NaCl roast followed by water leaching.

The suitability of the rotary kiln at Newport News, Va., for roasting Tasminex tungsten, ores was evaluated.

ROASTER FEED GRADE

Projection of the preliminary tests results for the Kara I samples gives the following ore processing scheme:

1. About 25 percent of the Kara I ore body is oxidized ore which cannot be upgraded.

ROASTER FEED GRADE CONT'D

2. Another 25 percent is oxidized ore of which an unknown proportion can be upgraded to 2.5 percent WO_3 by a wet magnetic separation.

3. The remaining 50 percent is unoxidized ore which can be upgraded to at least 10 percent WO_3 with a 90 percent recovery using a wet magnetic separation and flotation.

Table I shows that the average grade of the roaster feed ranges from 1.12 to 1.77 percent WO_3 depending on the average ore grade and on the proportion of the oxidized ore that can be upgraded by a wet magnetic separation. The average grade of all the drilling samples to date is 0.65 percent WO_3. Also, since upgrading the oxidized ore may not be practical, the most probable roaster feed grade was taken to be 1.2 percent WO_3.

Depending on the mining plan, it may not be feasible to blend the ore fractions as indicated.

EFFECT OF FEED GRADE

The roaster feed rate calculations were based on the following:

Plant Capacity	=	200,000 units/year
Onstream Time	=	90 percent
Overall WO_3 Recovery (includes upgrading)	=	85 percent
Instantaneous Feed Rate	=	600 lb.WO_3/Hr
	=	30 Units/Hr

The required rotary kiln ore feed rates for various feed grades are as follows:

Feed Grade % WO_3	Ore Feed Rate Tons/Hr.
0.5	60
0.75	40
1.0	30
1.2	25
1.5	20
2.0	15

115

Mr. R.J. Klotzbach
May 11, 1972

- 662117

ROTARY KILN CAPACITY

The operation of the following rotary kilns were compared:

	Location		
	Rifle	South Africa	Newport News
Shell Outside Diameter, ft.	9	9	9
Belly Outside Diameter, ft.	11	None	11
Length of Belly Section, ft.	35		17
Overall Length, ft.	360	180	120

Figure 1 shows the effect of kiln length on the capacity for rotary kilns of the same diameter. The slope of the line for Union Carbide's roasting operations is between those for wet and dry cement kilns. The capacity of the Newport News kiln is estimated to be about 11 tons of ore feed per hour. Preliminary sizing indicates that a 12 foot shell diameter by 200 foot long rotary kiln will be required to process the required 25 tons per hour.

T. J. Kagetsu

T.J. Kagetsu/cl

TABLE I

ROASTER FEED GRADE CALCULATIONS

Ore Type	Run of Mine Ore			Unoxidized Ore Upgraded			Unoxidized Ore and Half Oxidized Ore Upgraded		
	Distribution lb. ore	Grade % WO_3	Wt. WO_3 lb.	Distribution lb.	Grade % WO_3	WT WO_3 lb.	Distribution lb.	Grade % WO_3	Wt. WO_3 lb.
Oxidized	25	0.8	0.2	25	0.8	0.2	25	0.8	0.2
Oxidized	25	0.8	0.2	25	0.8	0.2	7.2*	2.5	0.18
Unoxidized	50	0.4	0.2	1.8	10	0.18	1.8	10.0	0.18
Composite	100	0.6	0.6	0.6	51.8	1.12	34	1.65	0.56
Oxidized	25	0.88	0.22	25	0.88	0.22	25	0.88	0.22
Oxidized	25	0.88	0.22	25	0.88	0.22	8*	2.5	0.20
Unoxidized	50	0.44	0.22	2	10	0.20	2	10.0	0.20
Composite	100	0.66	0.66	52	1.23	0.64	35	1.77	0.62

* 25 percent of the ore body upgraded to 2.5% WO_3 by a wet magnetic separation

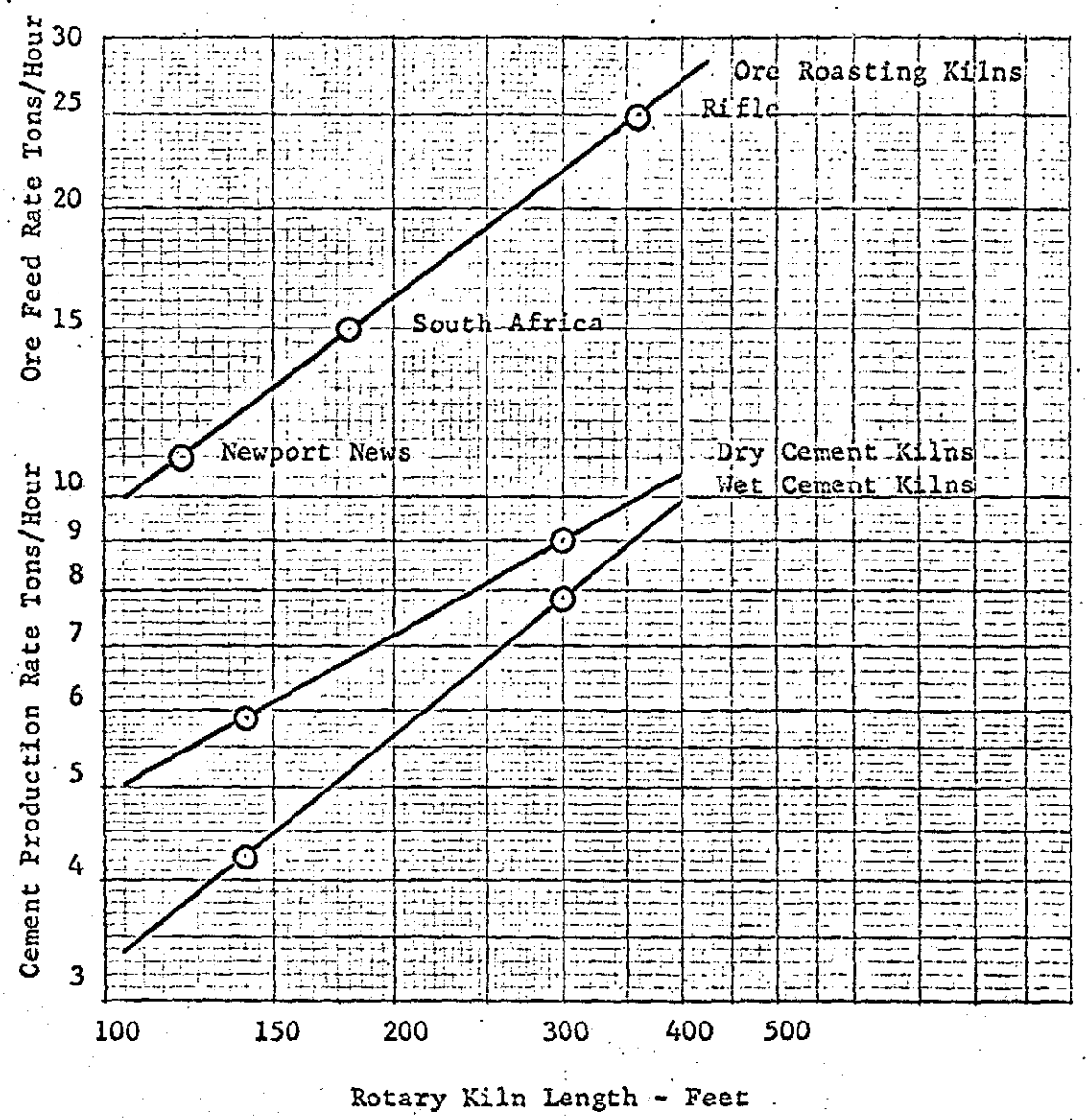

Ore Feed Rate Tons/Hour
Cement Production Rate Tons/Hour
30
25
20
15
10
9
8
7
6
5
4
3
100
150
200
300
400
500
Rotary Kiln Length - Feet
Ore Roasting Kilns
Rifle
South Africa
Newport News
Dry Cement Kilns
Wet Cement Kilns

FIGURE I

5 cm

INTERNAL CORRESPONDENCE 662120

MINING AND METALS DIVISION • P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Avenue
New York, New York 10017

Date May 4, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex Ore Samples

Copy to Messrs. A. W. Heuck (2)
J. C. Stephenson
M. Stern

Dear Lew:

Attached are our most recent salt roasting results on Tasminex ore samples. These data show that 90-98% of the WO_3 is solubilized in a 30-minute roast using equal portions of sodium carbonate and sodium chloride. Roasting with either salt alone proved unsatisfactory and finer grinding appeared to have a small effect on increased WO_3 recovery.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

RECEIVED
U. C. EXPLORATION CORP.
MAY 8 1972

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662121

Research and Development Department
Mining and Metals Division - UCC
Niagara Falls, New York
May 2, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. W. B. DeAtley
N. L. Grauerholz
D. J. Hansen

FROM: J. S. Fox

SUBJECT: Recovery of Tungsten Values from Tasminex Tungsten Ore Samples

SUMMARY

Tests were made to improve the roast-leach process for extracting tungsten values from air-dried (~25°C.) samples of Tasminex ores or concentrates. This was done by studying the effects of the variables of roasting and leaching on the amount of tungsten values solubilized in the subsequent water leach.

1. Effect of Time of Roasting

Adequate tungsten solubilization of 95% was obtained in a 1-hour roast of a trench sample (B1-B6) containing 1.38% WO_3 using 12.5% Na_2CO_3 along with 12.5% NaCl. This is equivalent to 9.1 pounds of Na_2CO_3 and 9.1 pounds of NaCl per pound of WO_3 in the sample. Increasing the roasting time to 3 hours slightly improved the WO_3 solubilization to 97%.

Adequate WO_3 solubilization (95%) was obtained on the total composite sample in 0.5 hour using the above amounts of Na_2CO_3 and NaCl. Increasing the roasting time to 3 hours did not further improve the WO_3 solubilization (95%).

Using the equivalent of 10.3 pounds of Na_2CO_3 and 10.3 pounds of NaCl per pound of WO_3 in a roast of the non-magnetic portion of the total composite sample, 98.5% WO_3 solubilization was obtained. Increasing the roasting time to 3 hours did not further improve the WO_3 solubilization.

All of the above results show that the air-dried (~25°C.) tungsten samples received to date can be roasted in a relatively short time and still obtain excellent WO_3 solubilizations.

2. Effect of Leaching Time

Tests were made on leaching of calcines from the roast of a trench (B1-B6) composite using 12.5% Na_2CO_3 and 12.5% NaCl in a roast at 800°C. One-hour leaches using approximately 4 ml. of water per gram of original ore gave

the same WO_3 solubilization (95%) as did the 3-hour leaches. These tests show that the tungsten values can be readily solubilized with a leaching time of no more than one hour.

3. Effect of Particle Size

Roasts were made on samples of a trench composite of surface and subsurface ore samples ground to different particle sizes. Adequate WO_3 solubilization (92%) was obtained from roasts at 800°C. using 12.5% Na_2CO_3 and 12.5% NaCl on ore ground to pass a minus 8 mesh Tyler screen, while 94% solubilization was obtained in a comparable roasted sample ground to minus 48 mesh. Further grinding to minus 200 mesh resulted in a WO_3 solubilization of 95%. These results show that adequate WO_3 solubilization in this sample can be obtained without excessive amounts of grinding.

4. Effect of NaCl Alone as a Roasting Reagent on Total Composite (Surface and Subsurface)

Inadequate WO_3 solubilization (less than 20%) was obtained in roasts of the total composite sample (1.21% WO_3) using 13 to 33% NaCl in the 1-hour roast at 800°C.

Effect of Na_2CO_3 Alone as a Roasting Reagent on Total Composite Sample or Non-Magnetic Portion of Total Composite

From 80% to 87% WO_3 solubilization was obtained in roasts of total composite sample (1.21% WO_3) using 12% to 24% Na_2CO_3 in the 1-hour roast at 800°C.

With 18% to 24% Na_2CO_3, only 62% to 63% WO_3 solubilization was obtained in 1-hour 800°C. roasts of the non-magnetic portion of the total composite sample.

These tests show that Na_2CO_3 alone was not as desirable as a roasting reagent for solubilization of the tungsten values as when employed along with NaCl in roasts at 800°C.

Effect of Na_2CO_3 Combined with NaCl as a Roasting Reagent

In 800°C. roasts of either the air-dried (~25°C.) total composite sample or the air-dried (~25°C.) non-magnetic portion of the total composite sample, better WO_3 extractions were obtained in 800°C. roasts using Na_2CO_3 combined with an equal weight of NaCl than when either Na_2CO_3 or NaCl was employed alone based on an equivalent amount of Na in the reagent. Further tests will be made to check this statement using larger amounts of Na_2CO_3 on the air-dried (~25°C.) non-magnetic portion of the total composite sample.

DISCUSSION

Since the last memorandum on April 6, 1972, experimental work was continued on the recovery of tungsten values from air-dried (~25°C.) Tasminex tungsten ore samples submitted by Mr. N. L. Grauerholz. The samples used were analyzed and described in the previous memorandum to Mr. R. G. Woolery of the

above date. A study was made on (1) the effect of the roasting time on the subsequent solubilization of the tungsten values in the water leach, (2) the effect of the leaching time of the roasted calcines, (3) the effect of the particle size prior to roasting on the subsequent tungsten solubilization, and (4) the effect of roasting with NaCl or Na_2CO_3 alone or NaCl along with Na_2CO_3 on the subsequent tungsten solubilization of the total composite sample or the magnetic fraction of the total composite sample.

1. Effect of Roasting Time

Samples of air-dried (~25°C.) Tasminex trench composite of B1-B6 (surface ore containing 1.38% WO_3 minus 200 mesh) were each roasted for 1, 2, or 3 hours with Na_2CO_3 combined with NaCl. The procedure used was the same as that previously described in the last memorandum of April 6, 1972 to Mr. R. G. Woolery. The conditions and results of these tests are shown in Table I (Tests 1, 2, and 3). These results show that 95% of the tungsten was solubilized in roasts with a residence time of only 1 hour at 800°C. and 97% in 3-hour roasts. These tests show that adequate tungsten solubilization can be achieved in a relatively short roasting period of no more than 1 hour.

Samples of the air-dried (~25°C.) total composite of Tasminex ore containing 1.21% WO_3 (surface and subsurface ores and minus 200 mesh powder) were each roasted for 0.5, 1.0, 2.0, or 3.0 hours at 800°C. with Na_2CO_3 combined with NaCl. The conditions and results of these tests are also shown in Table I (Tests 4, 5, 6, and 7). These results show that 95% of the tungsten values was solubilized in the 30-minute roast and 95% in the 3-hour roast. These results show that this total composite sample can be easily roasted with Na_2CO_3 combined with NaCl to obtain excellent tungsten solubilization in a relatively short roasting period without jeopardizing the tungsten solubilization. Tests using shorter roasting periods are planned.

Samples of the non-magnetic portion of the total composite ore sample (surface and subsurface minus 200 mesh air-dried powder) were each roasted for 1 or 3 hours with Na_2CO_3 combined with NaCl. The conditions and results of these tests are shown in Table I (Tests 8 and 9). In 1 hour, 99% of the tungsten was solubilized compared to 95% in the 3-hour roast. Again, the results show that good tungsten solubilization can be achieved in a relatively short roasting period.

2. Effect of Water Leaching Time

Tests were made to determine the effect of the leaching time on the amount of tungsten values solubilized from a roast of trench composite B1-B6 (surface ore 1.38% WO_3) using 12.5% Na_2CO_3 along with 12.5% NaCl. The conditions and results of these tests are shown in Table II. These results show that leaching the calcines for no more than 1 hour will solubilize (95% WO_3) as much of the tungsten values as would a leach of 3 hours' duration.

3. Effect of Particle Size

A sample of a composite B1-B6 (surface ore) was obtained which was ground to minus 8 mesh (Tyler). A split of the minus 8 mesh sample was ground to minus 200 mesh. A split of the minus 8 mesh was screened through a 48 mesh screen and the oversize was ground to minus 48 mesh. Each sample was given a wet screen analysis. These results are shown in Table III.

One-hour roast tests at 800°C. were made on each sample using 12.5% Na_2CO_3 along with 12.5% NaCl as the roasting reagent. The conditions and results of these tests are shown in Table IV. These results show that 92% WO_3 solubilization was obtained on the minus 8 mesh material, 94% on the minus 48 mesh material, and 95% on the minus 200 mesh material. These results show that the mesh sizes of the three samples tested indicate that the particle size difference in these samples did not cause but minor differences in the amount of tungsten solubilized. However, it should be noted that while grinding the ore to minus 8 mesh a considerable amount of minus 200 mesh material was obtained (approximately 30%), while 45% of minus 200 mesh material was found in the minus 48 mesh sample.

4. Effect of NaCl, Na_2CO_3, or NaCl Combined with NaCl:

Only NaCl in Roast of Total Composite Sample

A few roast tests were made on a composite of surface and subsurface tungsten ores (1.21% WO_3) using only NaCl as the roasting reagent in amounts ranging from 13% to 33% of the ore weight. These roast tests were made with a residence time of 1 hour at 800°C. The conditions and results of these tests are shown in Table V (Tests 1, 2, and 3). These results show that insufficient amounts of tungsten values were solubilized in any of the tests (less than 20% WO_3 solubilized).

Only Na_2CO_3 in Roast of Total Composite Sample

Two tests were made on the above-mentioned total composite sample using from approximately 12% to approximately 24% Na_2CO_3 as the roasting reagent. The roasting time and temperature were the same as the tests described above. The conditions and results of these tests are shown in Table V (Tests 4 and 5). These results show that 80% WO_3 extraction was obtained in the test using 11.9% Na_2CO_3 and 87% in the test using 23.9% Na_2CO_3. When the roasting time was extended to 3 hours in roasts using either 11.9% Na_2CO_3 or 23.9% Na_2CO_3, no further improvement in WO_3 solubilization was obtained. The conditions and results of these tests are shown in Table V (Tests 6 and 7).

Na_2CO_3 Combined with NaCl in Roast of Total Composite Sample

Two roast tests were made on the total composite ore sample (1.21% WO_3) used in the above tests. In these 800°C. roast tests 12.5% Na_2CO_3 combined with 12.5% NaCl was employed. One test had a residence time of 1 hour and the other test 3 hours. The conditions and results of these tests are shown in Table V (Tests 8 and 9). These tests show that 90% WO_3 extraction was obtained in the

1-hour roast while 95% WO_3 solubilization was obtained in the 3-hour test. These tests indicate that a combination of Na_2CO_3 and NaCl is better than roasting with either reagent separately. This can be seen in Figure 1, where the equivalent amount of Na in each reagent or combination of reagents is plotted versus the WO_3 extraction. It is much more advantageous economically to substitute part of the Na_2CO_3 with a cheaper reagent such as NaCl. Further tests will be made to determine if Na_2SO_4 can be used instead of NaCl in the combined reagent roasts since Na_2SO_4 is cheaper than Na_2CO_3 and is less corrosive and is less of a pollutant than NaCl.

Na_2CO_3 Alone in Roast of Non-Magnetic Fraction of Total Composite Sample

To further check the effect of the addition of Na_2CO_3 alone as compared to the addition of Na_2CO_3 along with NaCl in the roast, the following tests were made on the non-magnetic fraction of the total composite of Tasminex ore.

Two 1-hour tests were made at a roasting temperature of 800°C. using 18% and 24% Na_2CO_3, respectively. The conditions and results of these tests are shown in Table V (Tests 10 and 11 and Figure 1). These tests show that inadequate amounts of the tungsten values were solubilized in these roasts (62%-63% WO_3 solubilization).

Na_2CO_3 Combined with NaCl in Roast of Non-Magnetic Fraction Total Composite Sample

Several tests were made in which the non-magnetic portion of the total composite sample was roasted with Na_2CO_3 combined with NaCl. The conditions and results of these tests are shown in Table V (Tests 12, 13, 14, and 15). These tests show that 98.5% of the tungsten values were solubilized with 24% Na_2CO_3 and 24% NaCl in 1-hour roasts. With equivalent amounts of Na, 73% WO_3 solubilization was obtained in a roast using Na_2CO_3 and NaCl ($\simeq$ 10.3% Na) compared to 62.4% WO_3 solubilization when Na_2CO_3 alone was used ($\simeq$ 10.3% Na). These results are also shown in Figure 1 and again show that on an equivalent Na basis alone, it is more advantageous to use Na_2CO_3 combined with NaCl than with Na_2CO_3 alone.

J. S. Fox

J. S. Fox

JSF:ms

TABLE I

Effect of Time of Roasting on WO_3 Solubilization of Various Tasminex Samples

Constant Roast Conditions: (800°C. (Time as shown (Wet air over powder (-200 M) (All samples - 200 mesh powder

Test No.	Sample Description	% WO_3 in Sample	Reagents Added to Roast: Na_2CO_3 % Based on Ore Wt.	Na_2CO_3 lb./lb. WO_3 in Ore	NaCl % Based on Ore Wt.	NaCl lb./lb. WO_3 in Ore	Time of Roast hr.	% WO_3 Ext.	WO_3 in Tails %	pH of Filt.	% WO_3 Material Balance
1	B-1 - B-6 Surface Comp.	1.38	12.5	9.1	12.5	9.1	1.0	95.1	0.07	12.5	92
2	B-1 - B-6 Surface Comp.	1.38	12.5	9.1	12.5	9.1	2.0	95.6	0.06	12.5	91
3	B-1 - B-6 Surface Comp.	1.38	12.5	9.1	12.5	9.1	3.0	97.0	0.05	12.8	95
4	Total Comp.*	1.21	12.5	10.3	12.5	10.3	0.5**	95.0	0.06	12.4	94
5	Total Comp.*	1.21	12.5	10.3	12.5	10.3	1.0	90.2	0.12	12.4	89
6	Total Comp.*	1.21	12.5	10.3	12.5	10.3	2.0	95.0	0.06	12.4	89
7	Total Comp.*	1.21	12.5	10.3	12.5	10.3	3.0	95.0	0.06	12.4	91
8	Non-Mag Portion of Total Comp.	2.34	24.0	10.3	24.0	10.3	1.0	98.5	0.04	12.5	87
9	Non-Mag Portion of Total Comp.	2.34	24.0	10.3	24.0	10.3	3.0	95.0	0.13	12.2	90

*Total composite contains surface and subsurface Tasminex tungsten ores
**14 minutes to reach temperature of 800°C.

TABLE II

Effect of Time of Water Leaching of Na_2CO_3-NaCl Roasted Calcines

Roasting Conditions: (Trench Comp. B1 - B-6 (1.38% WO_3)
(12.5% Na_2CO_3
(12.5% NaCl
(800° C.
(1 hour - wet air

Leach Conditions: (4 ml. H_2O/g. ore
(B.P. under reflux
(Water washed
(Time as shown

Test No.	Leaching Time hr.	% WO_3 Extracted	% WO_3 in Tails	pH of Filt.	WO_3 Material Balance
1	1	95.1	0.07	12.5	92
2	3	94.9	0.07	12.5	93

TABLE III

Screen Analyses of Tasminex Composite of Surface Trench Samples

Screen Size Tyler (Wet Screening)	Sample A (-8 Mesh)	Sample B (-48 Mesh)	Sample C (-200 Mesh)
-8 + 10	3.63	-	-
-10 + 14	7.42	-	-
-14 + 20	8.98	-	-
-20 + 28	9.77	-	-
-28 + 35	8.61	-	-
-35 + 48	8.51	-	-
-48 + 65	5.98	19.67	-
-65 + 100	6.43	16.18	-
-100 + 150	5.47	11.16	-
-150 + 200	4.81	8.50	0.81
-200 + 270	3.09	5.96	5.97
-270 + 400	3.60	5.18	18.77
-400	23.70	33.35	74.45

TABLE IV

Effect of Particle Size of Tasminex Trench Sample Composite in Roast on Subsequent Solubilization of Tungsten Values in Water Leach

Roast Conditions: (Trench Comp. B1 - B6 (1.38% WO_3)
(1 hr.
(800° C.
(12.5% Na_2CO_3 ≏ 9.1 #/# WO_3 in ore
(12.5% NaCl ≏ 9.1 #/# WO_3 in ore

Test No.	Mesh Size (Tyler)(Wet Screening)	% WO_3 Extraction	% WO_3 in Tails	pH of Filt.	Material Balance, %
1	-8 Mesh (See Table IV, Sample A)	92.2	0.11	12.5	90
2	-48 Mesh (See Table IV, Sample B)	94.3	0.08	12.4	91
3	-200 Mesh (See Table IV, Sample C)	95.1	0.07	12.5	92

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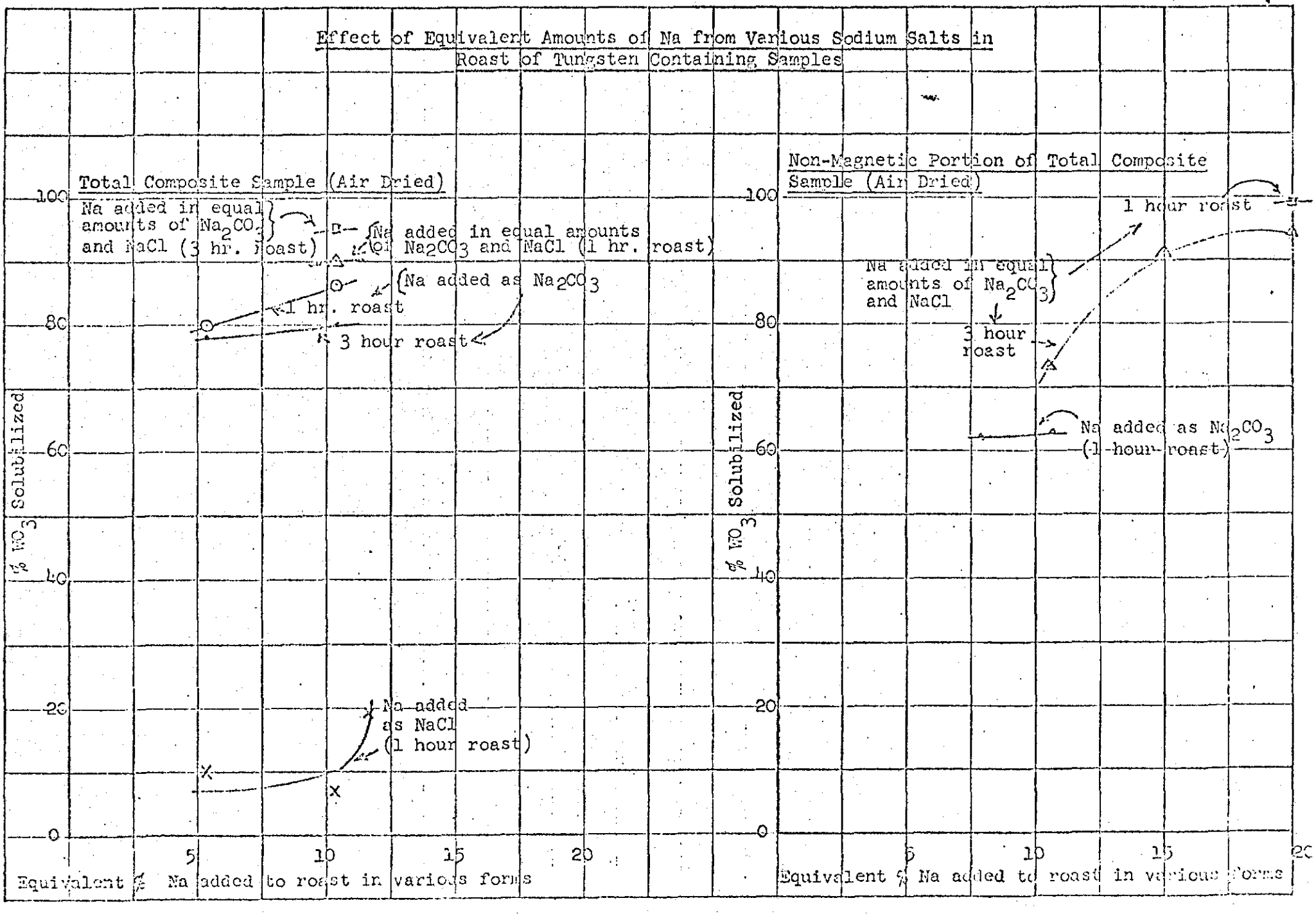
TABLE V

Effect of NaCl, Na_2CO_3, or NaCl Combined with Na_2CO_3 In Roast of Tasminex Tungsten Ore Samples on Solubilization of Tungsten Values in Subsequent Water Leach

Constant Roast Conditions: (-200 mesh powder, (800°C., (Wet air over bed

Test No.	Sample Tested	% WO_3 in Sample	Roast Time hr.	Reagents Added to Roast: NaCl %	NaCl #/#WO_3 in Ore	Na_2CO_3 %	Na_2CO_3 #/#WO_3 in Ore	Total ≠ Na Added to Roast %	% WO_3 Extracted	WO_3 in Tails %	pH of Filt.	WO_3 Material Bal. %
	Effect of NaCl alone (Total Composite)											
1	Surface & Subsurface or Total Comp.	1.21	1.0	13.2	10.9	0.0	0.0	5.2	10.0	1.10	8.0	103
2	" " " " "	1.21	1.0	26.3	21.7	0.0	0.0	10.3	6.9	1.10	7.5	96
3	" " " " "	1.21	1.0	33.2	27.4	0.0	0.0	13.0	19.3	0.95	7.7	96
	Effect of Na_2CO_3 alone (Total Composite)											
4	" " " "	1.21	1	0.0	0.0	11.9	9.8	5.2	79.7	0.25	12.2	93
5	" " " "	1.21	1	0.0	0.0	23.9	19.8	10.3	86.6	0.16	12.9	94
6	" " " "	1.21	3	0.0	0.0	11.9	9.8	5.2	78.0	0.27	12.0	99
7	" " " "	1.21	3	0.0	0.0	23.9	19.8	10.3	80.2	0.24	12.9	96
	Effect of NaCl Combined with Na_2CO_3 (Total Composite)											
8	" " " " " "	1.21	1	12.5	10.3	12.5	10.3	10.3	90.2	0.12	12.4	89
9	" " " " " "	1.21	3	12.5	10.3	12.5	10.3	10.3	95.0	0.06	12.4	91
	Effect of Na_2CO_3 alone (Non-Magnetic Portion of Total Composite)											
10	Non-Mag Portion of Total Comp.	2.34	1	0.0	0.0	18.0	7.7	7.81	62.4	1.02	12.6	98
11	" " " " "	2.34	1	0.0	0.0	24.0	10.3	10.4	62.7	1.01	12.8	105
	Effect of NaCl Combined with Na_2CO_3 (Non-Magnetic Portion of Total Composite)											
12	" " " " " "	2.34	3	12.5	5.3	12.5	5.3	10.3	73.2	0.72	10.3	100
13	" " " " " "	2.34	3	18.0	7.7	18.0	7.7	14.9	91.5	0.23	12.2	89
14	" " " " " "	2.34	1	24.0	10.3	24.0	10.3	19.8	98.5	0.04	12.5	87
15	" " " " " "	2.34	3	24.0	10.3	24.0	10.3	19.8	95.0	0.13	12.2	90

FIGURE 1



128 File - Tasminex - Kara

662130

INTERNAL CORRESPONDENCE

METALS AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. P. Twichell
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Ave.
New York, New York 10017

Date May 3, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex, Kara #1
Mineralogical Studies

Copy to Messrs. A. W. Heuck (2)
J. C. Stephenson
M. Stern

Dear Lew:

Attached is the present status of our mineralogical work on the six trench samples from Tasminex, Kara #1.

I believe the important significance of this work is that Ed has found what might be a correlation between the oxidized zone at higher elevation and that of lower elevation that could explain the difference in amenability of the two areas to magnetic beneficiation. If you remember from Norm Grauerholz's work, he obtained good beneficiation results from Trench 6 where as in Trench 1 through 5 from atop the hill upgrading was only marginal at best.

Ed has observed that in the higher elevation there is a predominance of goethite (poor amenability) to an almost total lack of goethite at the lower elevation (good amenability). He has speculated that this reflects a higher degree of laterization at the higher elevation. Because we have looked at only one sample from the lower elevation, it is difficult to place too much confidence in this hypothesis. Therefore, if we are to receive additional samples from Tasminex, we would appreciate the cooperation of the geologist in obtaining samples that would be meaningful in proving or disproving Ed's theory.

The occurrence of tungsten in the enriched oxidized zone is still under investigation. The non-scheelite tungsten is assumed to be present as vagrant ions in one or more of the gangue minerals. Evidence thus far indicates the most likely host is goethite, and additional studies are planned in an attempt to confirm these assumptions.

Very truly yours,

R. G. Woolery

RGW/bsn
Attach.

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662131

Research & Development Department
Mining & Metals Division - UCC
Niagara Falls, New York
April 26, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: R. & D. File
Messrs. N. L. Grauerholz
D. J. Hansen

FROM: E. E. Anderson

SUBJECT: Mineralogy of Tasminex Tungsten Samples

Studies of the nine samples from the Tasminex, Kara No. 1 tungsten deposit employing physical, chemical, X-ray, and optical techniques continued during the month of March in an effort to determine the mineralogy and to interpret this information in terms of amenability variations and extent and distribution of ore facies. Samples B1 and B6 were chosen initially for detailed mineralogic study because they appeared to represent two distinct types of near surface material from the trenches. Detailed studies of the other samples have also been started.

Interpretation of X-ray diffraction analyses of the six trench and three drill hole samples from the Tasminex project is summarized in Table I. It will be seen that these samples vary somewhat in degree of oxidation and hydration as witnessed by the relative abundances of magnetite (Fe_3O_4), hematite (Fe_2O_3), and goethite ($HFeO_2$). The other minerals recognized in the X-ray patterns of the head samples were diopside ($CaMgSi_2O_6$), vesuvianite [$Ca_{10}(Mg,Fe)_2Al_4Si_9O_{34}(OH)_4$], and scheelite? ($CaWO_4$) in the deepest core sample; and gibbsite[$Al(OH)_3$], quartz (SiO_2) and clay minerals in the near surface samples.

Tables II through IV show the relative abundance of the mineral constituents and tungsten concentration in various gravity (heavy liquid) and magnetic (Frantz) fractions from Samples B1 and B6.

Studies of density and magnetic fractions permit the identification of minerals not sufficiently abundant in the head samples to be detected by X-ray diffraction. In this manner the following minerals and/or occurrences have been noted. Scheelite and/or stoltzite ($PbWO_4$) are present to some degree in all samples but account for most or all of the tungsten only in Samples B6 and B102/3. It is interesting that these same two samples contain almost no goethite. Traces of goethite are detectable only in selected fractions of these samples subjected to heavy liquid and high-intensity magnetic treatments. Similarly, the hydrated garnet mineral hibschite ($3CaO \cdot Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$) and epidote [$Ca_2(Al,Fe)_2OHSi_3Al_{12}$] were detected in several high-intensity magnetic fractions of a methylene iodide sink product from Sample B6 (see Table IV).

Correlation of the mineralogic composition, sample location, and elevation of the near surface samples suggests that we are dealing with a classic example of the process of laterization. In this process the original calcium-iron-aluminum-silicate minerals are progressively altered to various clay minerals and then to hematite, goethite, and gibbsite. Similarly, the magnetite is altered to hematite and goethite. This is reflected chemically by progressive depletion of calcium and silicon coupled with a residual concentration of aluminum and iron. Departures from an ideal laterization sequence are present and can most likely be explained by variations in drainage and/or original composition. In the case of the Kara property, the laterization process also residually concentrated the tungsten. Since the amounts of scheelite and stoltzite detected in the near surface samples are insufficient to account for the tungsten present and since no additional tungsten minerals have as yet been established, the most promising assumption at this time would be that the tungsten is present as vagrant ions in one or more of the "gangue" minerals. From the data presented in Table III, the most likely tungsten host would be goethite. This will be investigated further next month.

The shift from dominance of goethite at higher elevations (Samples B1, B2, B3, and B102/1) to dominance of magnetite down the hill or at depth (B4, B5, B6, B102/2, and B102/3) suggests questions as to the relative abundance and distribution of the high-grade secondary material and the primary scheelite ore. These preliminary studies and the maps and section provided by L. Wright indicate that the residually-enriched zone is about 50 feet thick in the vicinity of drill holes DD101 and 102 near the top of the hill. More drill hole samples should be studied to determine the distribution and thickness of the enriched zone elsewhere. It is feasible that the study of additional samples, particularly those from the primary zone, could result in the development of criteria for distinguishing between primary and secondary ore types.

The distribution of tungsten values in three density fractions and the hand magnetic fraction is shown in Table V. In this study the highly magnetic material, magnetite and locked non-magnetics, has been removed from two size fractions from each sample. The sized, non-magnetic fractions were first separated into float and sink products in acetylene tetrabromide at a specific gravity of

2.96. The sink products from the 2.96 liquid were further separated in methylene iodide at a specific gravity of 3.33. Table VI shows the portions of each sample reporting to the various fractions.

From Table V it is quite evident that the tungsten is most highly concentrated in the sink fractions at 3.33 of the non-magnetic fractions. These fractions are being investigated further and the outcome of these studies will be reported at a later date.

Study of polished thin sections from the core samples (B102/2 and B102/3) and a residual rock fragment from Sample B6 shows that the magnetite is partially oxidized to hematite and goethite in even the freshest of these (see Figure 1). Iron and copper sulfides are present in B102/2 and B102/3. Pyrite (FeS_2) occurs as minute anhedral crystals on the order of one micron in diameter, disseminated in some of the magnetite (see Figure 2). Larger pyrite crystals and intergrowths of chalcopyrite ($CuFeS_2$), covelite (CuS), chalcocite (Cu_2S), and bornite (Cu_5FeS_4) occasionally occur as inclusions in magnetite, but more commonly occur filling interstices between magnetite, scheelite, and silicate mineral grains (Figure 3). In B102/3 the minerals scheelite, magnetite, and the sulfides are commonly concentrated in lenses or bands in the silicate host rock (Figure 4). The scheelite pods are quite coarse, sometimes measuring as much as five millimeters in long dimension.

E. E. Anderson

EEA:dmp
Attach.

TABLE I

MINERALOGIC COMPOSITION OF TASMINEX HEAD SAMPLES AS DETERMINED BY X-RAY DIFFRACTION *

Sample #	Magnetite	Hematite	Goethite	Diopside	Vesuvianite	Gibbsite	Quartz	Clay 14A	Clay 7A
B1	moderate	major	minor	–	–	trace	trace	trace	–
B2	minor	minor	major	–	–	trace	trace	–	–
B3	moderate	major	minor	–	–	–	trace	–	–
B4	major	moderate	minor	–	–	trace?	trace	trace	–
B5	major	minor	minor	–	–	–	trace	–	–
B6	major	minor	–	–	–	–	–	trace	–
B102/1	major	minor	moderate	–	–	–	trace	trace	trace?
B102/2	major	minor	trace	–	–	–	trace	minor	–
B102/3	major	minor	–	minor	minor	–	–	–	–

* See text for additional minerals detected in subfractions.

TABLE II

MINERALOGIC COMPOSITION OF FRANTZ MAGNETIC FRACTIONS OF 2.96 g/cc FLOAT PRODUCT OF HAND NON-MAGNETIC, 65 X 100 M PORTION OF TASMINEX SAMPLE B1

Fraction	Gibbsite	Quartz	14Å Clay	7Å Clay	Spectrographic Tungsten
Magnetic @ 0.1A	minor	moderate	_	_	.06-.6
Magnetic @ 0.4A	minor	moderate	major	_	.08-.8
Magnetic @ 0.8A	minor	minor	major	major?	.1-1.
Non-Magnetic @ 0.8A	major	major	minor	minor?	.08-.8
65 X 100 M 2.96 Float					.8-8.

TABLE III

MINERALOGIC COMPOSITION OF FRANTZ MAGNETIC FRACTIONS OF 3.33 g/cc SINK PRODUCT OF HAND NON-MAGNETIC, 65 X 100 M PORTION OF TASMINEX SAMPLE B1

Fraction	Goethite	Hematite	Magnetite	Spectrographic Tungsten
65 X 100 M 3.33 Sink				M
2nd Hand Mag	major	minor	trace	M
Magnetic @ 0.4A	major	minor	trace	M
Magnetic @ 0.8A	major	minor	_	M
Non-Magnetic @ 0.8A	insufficient sample for analysis			

TABLE IV

MINERALOGIC COMPOSITION OF FRANTZ MAGNETIC FRACTIONS OF 3.33 g/cc SINK PRODUCT OF HAND NON-MAGNETIC PORTIONS OF TASMINEX SAMPLE B6

Fraction	Vesuvianite	Hibschite	Epidote	Scheelite	Goethite	Spectrographic Tungsten
65 X 100 M 3.33 Sink						M
Magnetic @ 0.4A	major	moderate	trace	minor	trace	2.-20.
Magnetic @ 1.2A	major	trace	minor	trace	-	.08-.8
Non-Magnetic @ 1.2A	-	-	-	major	-	M
100 X 200 M 3.33 Sink						M
Magnetic @ 0.4A	moderate	moderate	minor	trace	trace	1.-10.
Magnetic @ 0.8A	major	trace	minor	trace	-	.08-.8
Magnetic @ 1.2A	major	trace	minor	-	-	.08-.8
Non-Magnetic @ 1.2A	-	-	-	major	-	M

TABLE V

DISTRIBUTION OF SEMIQUANTITATIVE TUNGSTEN VALUES OF HAND MAGNETIC AND HAND NON-MAGNETIC DENSITY FRACTIONS

		Hand Non-Magnetic		
Sample #	Hand Magnetic	Float @ 2.96	Float @ 3.33	Sink @ 3.33
B1 65X100M	.8-8.	.8-8.	.8-8.	M
B1 100X200M	.8-8.	*	.8-8.	M
B2 100X200M	.3-3.	.04-.4	.8-8.	M
B2 200X325M	.4-4.	.04-.4	.8-8.	M
B3 100X200M	.1-1.	.08-.8	.8-8.	M
B3 200X325M	.8-8.	.08-.8	.8-8.	M
B4 100X200M	.8-8.	.8-8.	M	M
B4 200X325M	.4-4.	.1-1.	.4-4.	M
B5 100X200M	.2-2.	.4-4.	.8-8.	M
B5 200X325M	.4-4.	.4-4.	1.-10.	M
B6 65X100M	.04-.4	.1-1.	.08-.8	M
B6 100X200M	.02-.2	.1-1.	.03-.3	M
B102/1 100X200M	.08-.8	.08-.8	.08-.8	M
B102/1 200X325M	.4-4.	.08-.8	.08-.8	M
B102/2 100X200M	.1-1.	.08-.8	.8-8.	M
B102/2 200X325M	.4-4.	.08-.8	.8-8.	M
B102/3 100X200M	not detected	.008-.08	not detected	.8-8.
B102/3 200X325M	.01-.1	.008-.08	.008-.08	2.-20.

* This fraction subdivided on Frantz magnetic separator. See Table II

TABLE VI

% SAMPLE DISTRIBUTION BY FRACTION

Sample #	Hand Magnetic	Hand Non-Magnetic		
		Float @ 2.96	Float @ 3.33	Sink @ 3.33
B2 100X200M	24	28	14	58
B2 200X325M	64	24	14	62
B3 100X200M	11	14	17	69
B3 200X325M	18	24	13	63
B4 100X200M	16	19	22	59
B4 200X325M	21	22	20	59
B5 100X200M	14	13	17	70
B5 200X325M	24	18	26	56
B102/1 100X200M	10	22	43	35
B102/1 200X325M	15	35	26	39
B102/2 100X200M	10	41	13	46
B102/2 200X325M	13	41	19	41
B102/3 100X200M	19	12	30	58
B102/3 200X325M	27	17	29	54

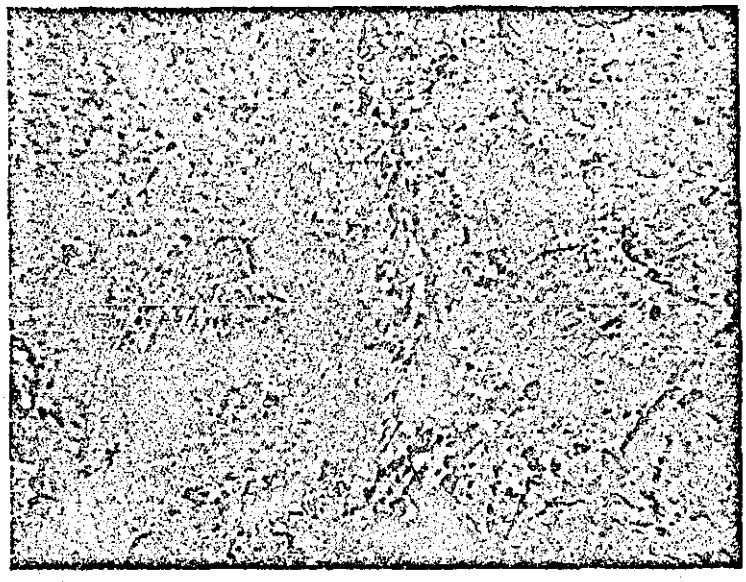

Figure 1. Photomicrograph of section of core from sample B102/3 showing partial alteration of magnetite (medium gray) by hematite and goethite (lighter gray). White spots are tiny pyrite crystals. Black areas are holes in polished section. Reflected light, 310X

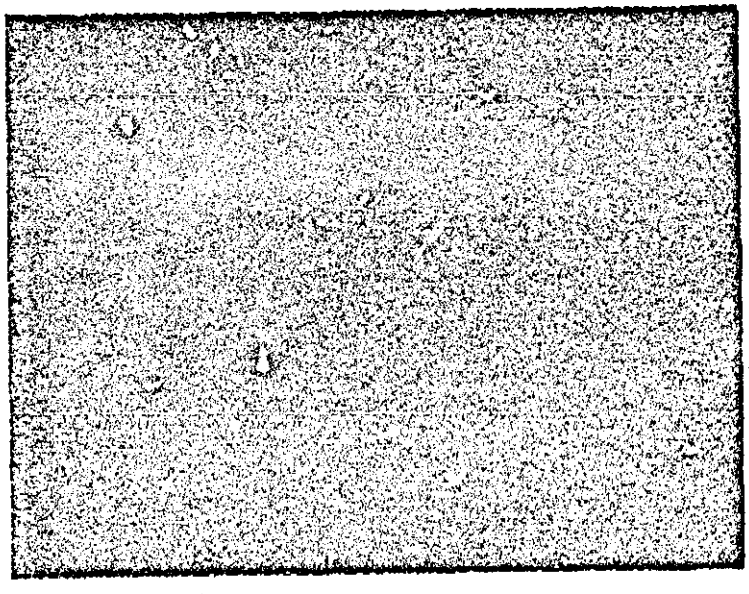

Figure 2. Photomicrograph of section of core from sample B102/3. Same minerals as Figure 1 showing detail of pyrite. Reflected light, 1000X

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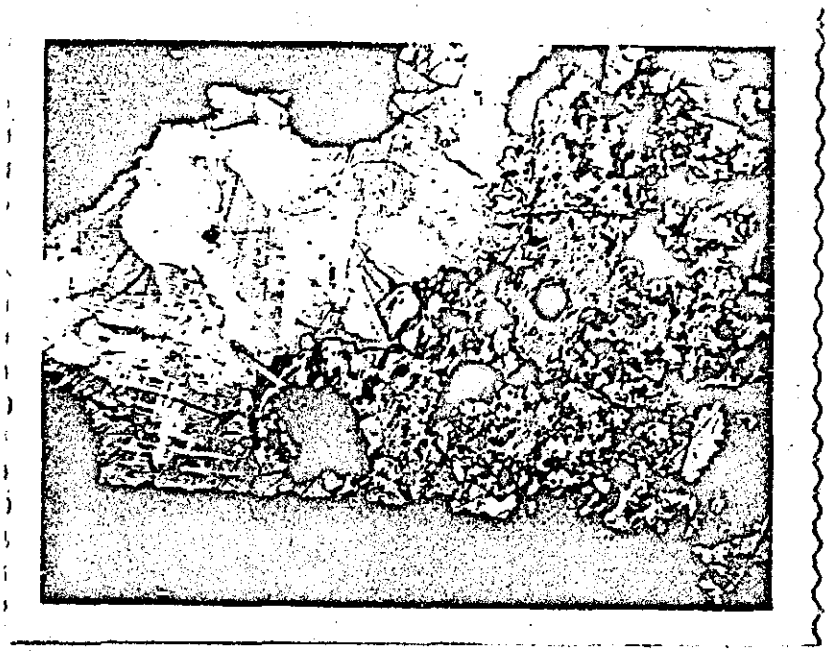

Figure 3. Photomicrograph of section of core from sample B102/3. Intergrowth of chalcopyrite (lightest gray) and bornite (dark medium gray) partially replaced by chalcocite (intermediate gray). Rough darker gray in corner is magnetite. Reflected light, 155X

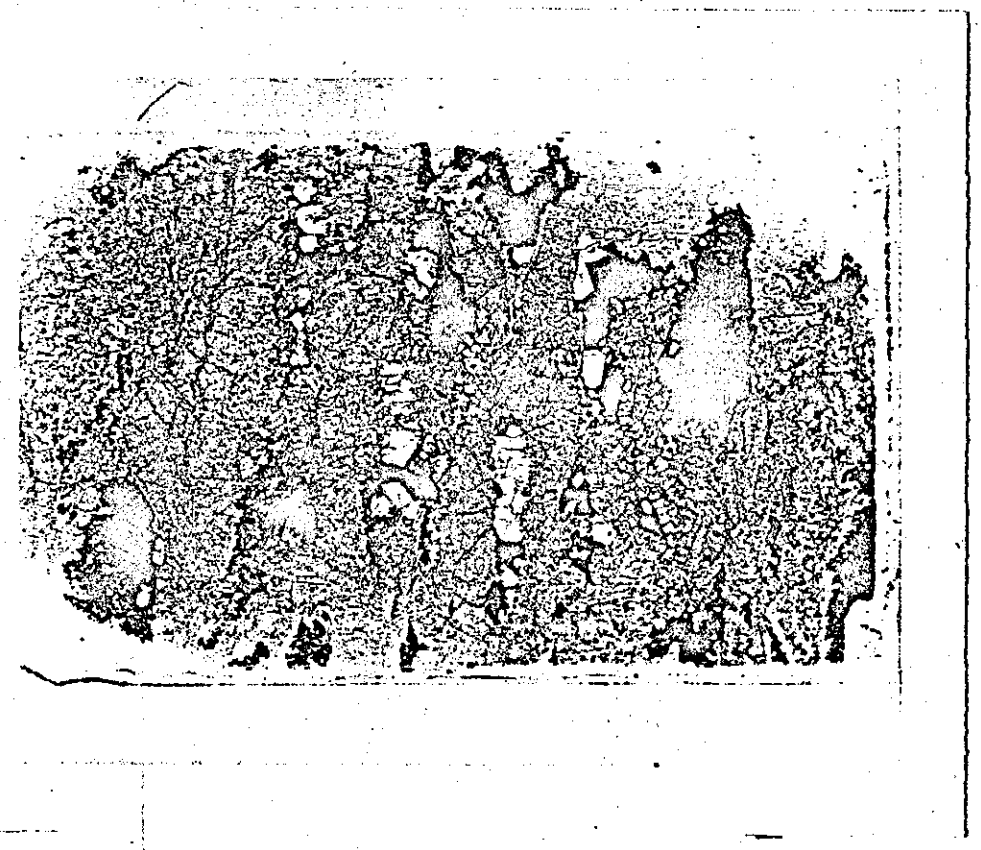

Figure 4. Photomicrograph of whole thin section of core from sample B102/3. Magnetite (black) - scheelite (white) pods and bands in silicate matrix (gray). Transmitted light, 2.7X

pp. File - Tungsten - Tasminex.

66214

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION

P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

(Name) Mr. L. P. Twichell
division 38th Floor, 270 Park Avenue
location New York, N. Y. 10017

Date April 21, 1972

Originating Dept. Research and Development Department

Answering letter date

copy to Mr. J. C. Stephenson
Dr. M. Stern
Mr. A. W. Heuck

Subject Tasminex Tungsten Processing
Progress During March, 1972

Dear Lew:

Efforts during March were concentrated on means of physically beneficiating the oxidized surface ore (trench samples) and also, the direct extraction of the tungsten values from untreated ore. The details are contained in the attached reports.

Magnetic separation of the oxidized samples resulted in a significant upgrading of the ore; however, the losses encountered may be prohibitive. A summary of the six test samples is as follows:

	% Grade - WO_3			% Distribution Non-Mags	
Sample No.	Heads	Mags	Non-Mags	Wt	WO_3
B-1 Trench 3 35- 60'	1.16	0.52	1.79	50.3	77.7
B-2 Trench 20 130-150'	1.32	0.34	2.05	69.4	92.1
B-3 Trench 16 20- 45'	1.51	0.58	2.41	50.7	81.1
B-4 Trench 18 80-105'	1.11	0.54	2.11	36.4	69.0
B-5 Trench 4 70-100'	1.76	0.58	3.94	35.1	78.7
B-6 Trench 13 500-520'	0.75	0.09	3.79	17.8	90.1

Obviously, on ores comparable to B-6 it is advantageous to upgrade by magnetic separation. B-2 would probably be satisfactory. The remaining samples are questionable. It is also apparent that the degree of oxidation is probably much less in sample B-6 as compared to the others.

In each of the above ores, a standard flotation was made on the non-magnetic fraction. In no case was the result sufficiently encouraging as to indicate any amenability to flotation.

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Some additional preliminary extraction tests were also completed during March. In all, three different routes were examined.

1. Na_2CO_3-NaCl roast at 800°C
2. NaOH leach
3. NaOH bake

Of the three only the Na_2CO_3-NaCl roast tests show promise. Tungsten extraction exceeding 90% was achieved with rather large concentration of salt. Indications are that from 300-700 pounds of salt per ton of ore are required depending, apparently, upon the grade of the feed. Preliminary estimates indicate a roast reagent cost of from $2.50 to $5.00 per stu WO_3 to obtain a 90%+ tungsten solubilization. In all cases the tungsten is recovered by a water leach.

Very truly yours,

R. G. Woolery

P.S. to A. W. Heuck
We are no longer sending any data to Dave Mathis. I assume if you want him to have this information you will forward it.

R.G.W.

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662143

Research & Development Department
P.O.Box 579, Niagara Falls, N.Y.
April 20, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

CC: Messrs. W. B. DeAtley
J. S. Fox
D. J. Hansen
R & D File

FROM: N. L. Grauerholz

SUBJECT: Recovery of Tungsten Values from Tasminex Tungsten Ore Samples

Summary

Flotation tests have been conducted on individual trench samples. The tests to date have not revealed any significant recovery or tungsten concentration in the flotation concentrate. It does not appear that any of the trench samples will be amenable to upgrading by flotation.

Composite samples and samples with the magnetite removed were given to the Hydrometallurgical Section for roast-leach tests. The results from these tests will be the subject of a separate report.

Discussion

Preliminary flotation tests were conducted on the nonmagnetic fraction from 1000-gram test charges of Surface Trench Samples B1-B6. The magnetic fraction was removed with a laboratory magnetic drum separator. The nonmagnetic fraction was ground to minus 100 mesh, with sodium carbonate and sodium silicate added to the grind. Sodium oleate was used as a collector rather than oleic acid. The rougher concentrate was cleaned by reflotation.

The magnetic separation-flotation tests on Trench Samples B1-B6 are shown in Table I through Table VI, attached. These results show that the magnetic iron fraction still carries substantial tungsten. None of these tests succeeded in producing a tungsten concentrate; however, B6, Trench 13, contained more scheelite and was more amenable to flotation.

The beneficiation of the ore by flotation has not been successful, but preliminary indications are that the ore can be treated by roasting and leaching. The success of the roast-leach process will depend on economics. Additional test work will be conducted toward this route. If the tungsten content of the magnetic fraction in this ore could be lowered, considerable upgrading could be accomplished and the roast-leach step would be more attractive. Additional tests are planned to determine if fine grinding will liberate the tungsten values from the magnetic fraction.

N. L. Grauerholz

N. L. Grauerholz

NLG:rvg
Attach.

TABLE I

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-1 Trench 3 35-60'

Sample: 1000 Grams - 8 Mesh Ore
Magnetic Iron Removed - Nonmags Ground for Flotation

	Min. Time	pH	Reagents, Lb./Ton of Original Ore			
			Na_2CO_3	Na_2SiO_3	Calgon	Sodium Oleate
Grind	5	7.6	0.4	0.4		
Cond.	3	9.1	2.0	0.4	0.4	
R. Float.	8					0.50 (Staged)
Cl. Flot.	3		0.4			0.25 (Staged)

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	49.7	0.52	22.3
R. Tail	47.4	1.72	70.2
Cl. Tail	1.5	2.55	3.3
Cl. Conc.	1.4	3.50	4.2
Calculated	100.0	1.16	100.0
Actual		1.55	

Remarks: No scheelite detected by U.V.

TABLE II

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-2 Trench 20 130-150'

Sample: 1000 Grams - 8 Mesh Ore
Magnetic Iron Removed - Nonmags Ground for Flotation

	Min. Time	pH	Reagents Lb./Ton of Original Ore Na_2CO_3	Na_2SiO_3	Sodium Oleate
Grind	5	7.6	0.4	0.4	
Cond.	3	8.0	0.4	1.2	
R. Flot.	12	8.0			0.6 (Staged)
Cl. Flot.	6	8.6		0.8	

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	30.6	0.34	7.9
R. Tail	47.8	1.65	59.7
Cl. Tail	18.3	2.07	28.7
Cl. Conc.	3.3	1.53	3.7
Calculated	100.0	1.32	100.0
Actual		1.37	

Remarks: No scheelite detected by U.V.

TABLE III

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-3 Trench 16 20-45'

Sample: 1000 Grams - 8 Mesh Ore
Magnetic Iron Removed - Nonmags Ground for Flotation

	Min. Time	pH	Reagents Lb./Ton of Original Ore Na_2CO_3	Na_2SiO_3	Sodium Oleate
Grind	5	8.3	0.4	0.4	
Cond.	3	8.3	0.8	1.2	
R. Flot.	12	8.3			0.75 (Staged)
Cl. Flot.	5	8.3		0.8	

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	49.3	0.58	18.9
R. Tail	19.7	2.78	36.2
Cl. Tail	28.6	2.22	42.0
Cl. Conc.	2.4	1.77	2.9
Calculated	100.0	1.51	100.0
Actual		1.61	

Remarks: No scheelite detected by U.V.

TABLE IV

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-4 Trench 18 80-105'

	Min. Time	pH	Reagents Lb./Ton Original Ore Na_2CO_3	Na_2SiO_3	Unitol DS4
Grind	5	8.0	1.0		
Cond. No. 1	3	8.1	2.0		
Cond. No. 2	3	9.3	3.0	1.5	
R. Flot.	10	9.3			0.25 (Staged)
Cl. Flot.	5	9.5		0.5	

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	63.61	0.54	31.05
R. Tail	33.53	1.91	57.90
Cl. Tail	2.50	2.08	4.70
Conc.	0.36	19.51	6.35
Calculated	100.00	1.11	100.00
Actual		1.29	

Remarks: No scheelite detected by U.V.

TABLE V

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-5 Trench 4 70-100'

	Min. Time	pH	Reagents, Lb./Ton Original Ore Na_2CO_3	Na_2SiO_3	Sodium Oleate
Grind	5	7.6	0.4		0.4
Cond.	3	8.2	0.4	1.2	
R. Flot.	10	8.2			0.75 (staged)
Cl. Flot.	8	9.1	0.25	0.75	0.20 (staged)

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	64.9	0.58	21.3
2nd Mag. Conc.(1)	1.7	1.27	1.2
R. Tail	24.0	4.06	55.0
Cl. Tail	9.1	4.22	21.6
Cl. Conc.	.3	4.78	0.9
Calculated	100.0	1.76	100.0
Actual		1.81	

(1) Magnetic Fraction removed after grinding original nonmags.

TABLE VI

Tasminex Amenability Test

Magnetic Separation-Flotation Concentration

B-6 Trench 13 500-520'

	Min. Time	pH	Reagents Lb./Ton of Original Ore Na_2CO_3	Na_2SiO_3	Sodium Oleate
Grind	5	7.8	0.4	0.4	
Cond.	3	8.2	0.4	1.2	
R. Flot.	12	8.2			0.8 (Staged)
Cl. Flot.	5	8.8	0.25	0.8	0.2 (Staged)

	% Wt.	% WO_3	Dist. WO_3
Mag. Conc.	82.2	0.09	9.9
2nd Mag. Conc.	0.9	0.52	0.7
R. Tail	11.7	0.67	10.5
Cl. Tail	1.3	2.35	4.0
Cl. Conc.	3.9	14.19	74.9
Calculated	100.0	0.75	100.0
Actual		0.68	

Remarks: Cleaned concentrate contains much scheelite

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662150

Research and Development Department
Niagara Falls, New York
April 6, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPIES: Messrs. W. B. DeAtley
N. L. Grauerholz
D. J. Hansen

FROM: J. S. Fox

SUBJECT: Recovery of Tungsten Values from Tasminex Tungsten Ore Samples

SUMMARY

A few preliminary tests were made to extract the tungsten values from various Tasminex samples. Roasting tests were made using a mixture of Na_2CO_3-NaCl as the roasting reagent. NaOH leaching tests and a few NaOH bake tests were also made to solubilize the tungsten values.

Excellent tungsten extractions of 95% to 97% were obtained from a composite of either surface trench samples or a composite made of surface and subsurface samples by a 12-1/2% Na_2CO_3-12-1/2% NaCl roast at 800°C. The resulting solution from the surface ore composite did not contain any excessive amounts of objectionable impurities. The cost of chemicals to recover 95% to 97% of the tungsten values from each of these two composites would be about 20 to 23 cents per pound of extracted WO_3 based on Na_2CO_3 at $35 per ton and NaCl at $7.10 per ton.

Roast tests on the non-magnetic fraction of the subsurface ore (core sample) showed that 90% of the tungsten values could be solubilized by a 12-1/2% Na_2CO_3-12-1/2% NaCl roast. With 24% Na_2CO_3-24% NaCl in the roast, 98% of the tungsten values was solubilized in the water leach.

Roasts on the non-magnetic portion of the composite of the surface and subsurface ore showed that 92% of the tungsten values could be solubilized in an 18% Na_2CO_3-18% NaCl roast and 95% of the values could be solubilized using 24% Na_2CO_3-24% NaCl in the roast. The chemical cost to recover 95% of the tungsten values would be about 23 cents per pound of WO_3 extracted.

NaOH leach tests at atmospheric pressure and at the boiling point employing the equivalent of 50 to 400 pounds of NaOH per ton did not solubilize more than 34% of the tungsten values. This amount of solubilization is inadequate.

A few bake tests were made to solubilize the tungsten values. The surface ore composite was mixed with an NaOH solution and baked overnight at 242°C. The residue was water leached, and under conditions in which the equivalent of 50 pounds of NaOH per ton was employed, an inadequate extraction of 35% WO_3 was obtained. When the equivalent of 200 pounds of NaOH was employed per ton of ore, an extraction of 77% WO_3 was obtained. This bake procedure was not as attractive as the roast procedure for solubilization of the tungsten values.

DISCUSSION

Experimental work was performed to recover the tungsten values from Tasminex ore found in Australia.

This ore composite of equal weights of B-1, B-2, B-3, B-4, B-5, and B-6 contains 1.38% WO_3. The chemical and spectrographic analyses of this composite are shown in Table I.

The drill core sample taken at 102'-1-3' was magnetically treated to remove the magnetic materials. The non-magnetic portions contained 23.7% of the weight of the original core sample and 87.0% of the tungsten based on the separation made by Mr. N. L. Grauerholz. The non-magnetic concentrate contained 2.09% WO_3. The chemical and spectrographic analyses of this concentrate are also shown in Table I. The magnetic portion contained 0.10% WO_3.

A total composite was made containing equal weights of three core samples and 6 trench samples, B-1 through B-6. The chemical and spectrographic analyses of this total composite are shown in Table I.

This composite was treated to remove the magnetic materials by N. L. Grauerholz. The non-magnetic portion contained 38.7% of the weight and 83.6% of the WO_3. The non-magnetic portion contained 2.34% WO_3 by weight. The chemical and spectrographic analyses of this sample are shown in Table I. The magnetic portion contained 0.34% WO_3 by weight.

Experimental work on a previous tungsten ore sample received from Nevada called Golconda ore was successfully roasted with a mixture of Na_2CO_3 and NaCl so that approximately 87% of the WO_3 values in this Golconda ore was subsequently solubilized in the water leach. The successful roasting conditions used on this Golconda ore were applied to the present Tasminex ore samples. In addition to the roast tests, a simple NaOH leach at the boiling point was also attempted to solubilize the tungsten values. Finally, a couple of bake tests were made in which the ore was mixed with aqueous NaOH and baked overnight at 242°C. The residue was then water leached. Following is a brief description of the procedures and results obtained on the above tests.

I. Na_2CO_3-NaCl Roast Tests

Four samples of Tasminex ore material were each roasted with Na_2CO_3 mixed with NaCl at 800°C. for 3 hours with wet air passing over the thin (1/8") bed of powder (-200 mesh) mixed with the reagents in an Inconel X dish. The air-cooled calcines were water leached with 4 ml. of water per gram of original Tasminex sample. The mixture was filtered and water washed. The wet residue was oven dried and weighed, and chemically analyzed for WO_3. The measured total filtrate was also analyzed for WO_3. From these results and WO_3 in the starting materials, WO_3 solubilization was calculated. The results obtained are shown in Table II.

From 91% to 97% WO_3 solubilization was obtained on the composite of surface trench samples of Tasminex ore. The roast using 6.25% Na_2CO_3 and 10% NaCl yielded 91% WO_3 extraction at a chemical cost of approximately 12 cents per pound of WO_3 solubilized. The chemical costs of the roast using 12.5% Na_2CO_3 and 12.5% NaCl to solubilize 97% of the tungsten values would be about 21 cents per pound of WO_3 solubilized. These preliminary tests show that the tungsten values from the surface trench samples are amenable to the Na_2CO_3-NaCl roast and that the chemical costs are not exorbitant. The solutions from these roasts did not contain excessive amounts of objectionable impurities as shown in Table III.

A composite sample of 6 trench surface samples and 3 core samples (approximately 120') was roasted as described above. The conditions and results of this total composite sample are shown in Table II (Tests 3 and 4). These results show that 95% WO_3 extraction was obtained when 12.5% Na_2CO_3 and 12.5% NaCl were employed in the roast. When 6.25% Na_2CO_3 and 10% NaCl were employed, 86% WO_3 extraction was obtained. These tests show that the mixture of core samples and trench samples is also amenable to the Na_2CO_3-NaCl roast procedure.

A composite sample of core material was magnetically treated. The non-magnetic portion was roasted as described above. The conditions and results of these tests are shown in Table II (Tests 5 through 8). These results show that 90% of the WO_3 values was solubilized in this upgraded core sample of Tasminex ore when 18% Na_2CO_3-18% NaCl was used in the roast. With 24% Na_2CO_3-24% NaCl in the roast, 98% of the WO_3 values was solubilized.

The total composite sample of 6 surface ore samples and 3 core samples was magnetically treated to yield a non-magnetic portion of material which contained 2.34% WO_3. This sample was also roasted as described. The conditions and results of these tests are shown in Table II (Tests 9-12). These results show that 92% WO_3 extraction was obtained with a roast containing 18% Na_2CO_3 and 18% NaCl. With a roast containing 24% Na_2CO_3 and 24% NaCl, an extraction of 95% was obtained.

II. NaOH Leach Tests at Atmospheric Pressure

A series of NaOH leach tests were made to solubilize the tungsten values from Tasminex trench sample Composites B-1 through B-6. These leaches were made at the boiling point under reflux for 3 hours at approximately 20% pulp density (5 ml./g. ore) using the equivalent of 50 to 400 pounds of NaOH per ton of ore. After leaching, the mixtures were filtered and the residues were water washed. The oven dried residues were weighed and chemically analyzed for WO_3 and the measure filtrates were also analyzed for WO_3. From these results and the WO_3 in the starting sample, the WO_3 solubilization was calculated. The conditions and results obtained are shown in Table IV. These results show that poor solubilization of the tungsten values was obtained by NaOH leaching under the conditions employed.

III. NaOH Bake Tests

Two bake tests were made to determine if a mixture of Tasminex trench composite (B-1 to B-6) and NaOH solution heated to 242°C. overnight would solubilize the tungsten values.

In the first test, 40 grams of Tasminex ore composite (B-1 to B-6) were mixed with 10 ml. of NaOH solution containing 100 g./l. NaOH in an Inconel X dish. The mixture was placed in an oven set at approximately 242°C. overnight. The dried residue was water leached with 200 ml. of water for 3 hours at the boiling point under reflux. The mixture was filtered and water washed. The dried residue was weighed and analyzed for WO_3, and the total filtrate was measured and analyzed for WO_3. From these results, the WO_3 solubilized was calculated. Test 2 was performed as in Test 1 except more NaOH was added to the ore composite prior to baking.

The conditions and results of these two tests are shown in Table V. These results show that with the equivalent of 50 pounds of NaOH per ton of composite in the bake, 35% of the tungsten values was solubilized, while 77% WO_3 solubilization was obtained when the equivalent of 400 pounds of NaOH was employed.

In view of the success obtained with the Na_2CO_3-NaCl roasts, tests using this bake procedure were not pursued at this time since these bake tests did not achieve the desired tungsten solubilization.

J. S. Fox

JSF:ms

TABLE I

Chemical and Spectrographic Analyses
of Tasminex Ores and Concentrates

Sample:	Surface Composite B1-B6	Total Comp. of Core & Trench Samples	Non-Mag Portion of Core Comp.	Non-Mag Portion of Total Comp. (Core & Trench)
Chem. Anal.				
% WO_3	1.38	1.21	1.79	2.34
Spec. Anal.	Range in Per Cent			
Ag	-	-	0.0008-0.008	-
Al	0.8-8.0	0.8-8.0	1-10	0.8-8.0
B	-	0.002-0.02	0.02-0.2	0.002-0.02
Be	0.0008-0.008	-	0.002-0.02	0.001-0.01
Ba	-	0.0008-0.008	-	-
Bi	0.02-0.2	0.02-0.2	0.01-0.1	0.02-0.2
Ca	0.03-0.3	0.08-0.8	Major	0.08-0.8
Cr	-	0.003-0.03	0.002-0.02	0.002-0.02
Co	0.004-0.04	0.004-0.04	-	-
Cu	0.08-0.8	0.04-0.4	0.08-0.8	0.04-0.4
Fe	Major	Major	2-20	Major
Ga	0.002-0.02	0.002-0.02	0.002-0.02	0.002-0.02
Mg	0.08-0.8	0.08-0.8	0.8-8	0.08-0.8
Mn	0.08-0.8	0.08-0.8	0.02-0.2	0.08-0.8
Mo	0.004-0.04	0.004-0.04	0.008-0.08	0.008-0.08
Ni	-	0.0008-0.008	0.002-0.02	0.001-0.01
Pb	0.08-0.8	0.08-0.8	0.04-0.4	0.08-0.8
Si	0.2-2.0	0.2-2.0	Major	0.4-4
Sn	0.08-0.8	0.04-0.4	0.04-0.4	0.08-0.8
Sr	-	-	0.008-0.08	-
Ti	0.02-0.2	0.02-0.2	0.02-0.2	0.03-0.3
V	-	0.0008-0.008	0.002-0.02	0.002-0.02
W	See Above	See Above	See Above	See Above
Zn	0.02-0.2	0.01-0.1	0.008-0.08	0.008-0.08
Zr	-	-	-	0.002-0.02

TABLE II

Effect of Na_2CO_3-NaCl Roasting of Various Tasminex Tungsten Ore Samples and Ore Concentrates on Tungsten Solubilization

Test No.	Sample Description: Name	% WO_3: Calc. from Res. & Filt.	% WO_3: by Chem. Anal.	Na_2CO_3 Added: %	Na_2CO_3 Added: #/Ton	NaCl Added: %	NaCl Added: #/Ton	Based on Res. & Head	WO_3 Content of Tails %	pH of Filt.	% WO_3 Matl. Bal.
1	Composite of B1-B6 (surface ore)	1.30	1.38	6.25	125	10.0	200	91	0.12	10.8	96
2	Same as above	1.33	1.38	12.5	250	12.5	250	97	0.05	12.8	95
3	Composite of B1-B6 plus core samples (surface and subsurface ore)	1.14	1.21	6.25	125	10.0	200	86	0.17	11.1	94
4	Same as above	1.10	1.21	12.5	250	12.5	250	95	0.06	12.4	91
5	Non-magnetic portion of core sample composite (sub-surface ore conc.)	1.76	1.79	6.25	125	10.0	200	23	1.29	9.1	103
6	Same as above	1.77	1.79	12.5	250	12.5	250	83	0.27	10.4	99
7	Same as above	1.52	1.79	18.0	360	18.0	360	90	0.16	11.5	85
8	Same as above	1.66	1.79	24.0	480	24.0	480	98	0.03	11.7	92
9	Non-magnetic portion of total composite of surface and sub-surface ore	2.37	2.34	6.25	125	10.0	200	33	1.87	10.1	101
10	Same as above	2.34	2.34	12.5	250	12.5	250	73	0.72	10.3	100
11	Same as above	2.07	2.34	18.0	360	18.0	360	92	0.23	12.2	89
12	Same as above	2.11	2.34	24.0	480	24.0	480	95	0.13	12.2	90

TABLE III

Chemical and Spectrographic Analyses of Solutions from Na_2CO_3-NaCl Roasts of Tasminex Trench Composites B1 through B6

Chemical Analyses	Filtrate from Roast with 12-1/2% Na_2CO_3 and 12-1/2% NaCl	Filtrate from Roast with 6-1/4% Na_2CO_3 and 10% NaCl
g./l. WO_3	1.66	1.56
g./Cr	0.27	0.27
Spectrographic Analyses - Range		
Al	0.4-4.0	0.04-0.4
B	0.004-0.04	
Be	0.0002-0.002	
Cr	See above	See above
Ga	0.001-0.01	-
Mo	0.02-0.2	0.01-0.1
Na	Major	Major
Si	0.004-0.04	0.001-0.01
Sn	0.004-0.04	0.001-0.01
V	0.001-0.01	0.0004-0.004

TABLE IV

Effect of Atmospheric Pressure Leaching Tasminex Tungsten Ore at Boiling Point with NaOH on Tungsten Solubilization

Constant Leach Conditions: (20% P.D. (5 ml./g. ore)
(B.P.
(3 hr. under reflux
(Ore composites B1 through B6
((1.38% WO_3)

Test No.	NaOH Conc. in g./l.	NaOH #/Ton Ore	% WO_3 Solubilized	WO_3 in Tails %	pH of Filt.	% WO_3 Matl. Bal.
1	5	50	25.0	1.08	12.2	101
2	10	100	25.2	1.08	12.3	103
3	20	200	26.4	1.08	12.7	96
4	40	400	33.9	0.96	12.7	100

56 File - Tungsten + Tasmanex

662158

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION • P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

(Name) Dr. D. L. Mathias, Jr.
ision Australia & New Zealand
ation Exploration Co.
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Australia

Date March 14, 1972

Originating Dept. Research and Development Department

Answering letter date

Subject Tasminex Tungsten
Kara 1 Project

by to

Messrs. D. J. Hansen
A. W. Heuck/L. A. Wright
J. A. Stephenson
L. P. Twichell

Dear Dave:

I am enclosing our Progress Reports for February concerning the Tasminex tungsten samples. One is the mineralogical report and the other concerns the upgrading program.

We were quite successful in treating the unoxidized samples by flotation in that we achieved a reasonable grade (9+%) and an encouraging WO_3 recovery 80+%. This is only scope testing so the chances of improving on these results is good.

The oxidized material is going to prove more difficult as we expected. Our preliminary flotation test resulted in a high-grade concentrate (19.5%) but the WO_3 recovery was extremely low (~6%). It appears that the unaltered scheelite (or stolzite) floats quite well but the altered WO_3 mineral?? shows no response to our flotation scheme. My guess is that we will not be able to upgrade this oxidized material by any beneficiation technique.

We have also initiated some chemical treatment schemes for solubilizing the oxidized tungsten which are in progress but not reported as yet. These have been only scope-type testing and further determination of the variables is still under investigation. A summary of these results are:

Type of Processing	Reagents	Lb./Ton Added	% WO_3 Solubilized
800°C Roast	Na_2CO_3 + NaCl	125+200	91.4
" "	Na_2CO_3 + NaCl	250+250	96.5
Atmos. Leach	NaOH	50	25.0
" "	"	100	25.2
" "	"	200	26.4
" "	"	400	33.9
Caustic Bake 240°C	NaOH	50	35.0
" "	"	200	78.3

No attempt has been made to optimize condition as our objective is to scope the various techniques before examining any one approach thoroughly.

I will keep you up-to-date as to our progress by forwarding our Monthly Progress Reports. In the meantime, if you have any questions or comments, I would be glad to hear them.

Very truly yours,

R. G. Woolery

RGW/bsn
Encl.

SUMMARY ADDITION

The unoxidized material from Drill Hole 102/3 50-70' was subjected to a magnetic separation-gravity concentration step. The magnetic fraction was extremely low in tungsten indicating that an intimate association of tungsten and magnetite is not present in the deeper zones of the ore body. Gravity concentration did not produce a high-grade concentrate as other minerals present in the ore have a specific gravity close to the specific gravity of tungsten.

A preliminary flotation test was conducted on the nonmagnetic fraction from Drill Hole 102/3 50-70'. A cleaned concentrate was produced by flotation that assayed over 9% WO_3 and recovered over 80% of the total WO_3 in the sample.

A preliminary flotation test was also conducted on the nonmagnetic fraction from Sample B-4, Trench 18, 80-105 feet. A cleaned concentrate that assayed 19.51% WO_3 was produced. Recovery was only 6% of the total WO_3. The tungsten recovered was identified as being mostly stolzite rather than scheelite. The high losses of WO_3 were distributed throughout the sample revealing that the surface ore is much more refractory than the ore from the deeper zones.

SUBJECT: TASMINEX TUNGSTEN SAMPLES Job No. 845-94101

WORK BY: C. S. Thompson, E. E. Anderson, R. Labosky | PERIOD ENDING: February 29, 1972

DATA FILED IN: NB 2345 | Page 1 of 19

SUMMARY

Mineralogical studies of the Tasminex, Kara No. 1 tungsten deposit are continuing on six (6) trench and three (3) drill core samples. Analytical data are presented indicating that the tungsten in the highly oxidized surface material is primarily present in mineral phases other than scheelite ($CaWO_4$). Studies are in progress to determine the nature and distribution of these non-scheelite, tungsten-bearing phases. To date, the only other tungsten mineral identified has been stolzite ($PbWO_4$) and that in only one sample, a float concentrate. The majority of the tungsten in the surface material appears to be very fine grained and intimately intermixed or intergrown with the gangue minerals, particularly the magnetite and other iron oxides.

DISCUSSION

Mineralogical studies of the tungsten-bearing material from the Tasmanian, Kara No. 1 project are continuing on nine (9) samples received from the Australian and New Zealand Exploration Company, Burnie, Tasmania, Australia. These samples, listed below, represent material found in six (6) trenches and one (1) drill hole distributed at varying intervals covering some 1000 feet of the property.

Field No.	N.F. No.	Identification	
B1	2345-92-1	Trench 3	35- 60'
B2	2	Trench 20	130-150'
B3	3	Trench 16	20- 45'
B4	4	Trench 18	80-105'
B5	5	Trench 4	70-100'
B6	6	Trench 13	500-520'
B102/1	7	Drill Hole 102	0- 15'
B102/2	8	Drill Hole 102	15- 50'
B102/3	9	Drill Hole 102	50- 70'

The samples, as received, were examined and several mineralogical specimens from each were taken for microscopic examination. Thin sections and polished sections of these specimens are being prepared and will be described in the near future.

All samples were prepared for study by crushing to minus 8 mesh (Tyler series) as described by N. L. Grauerholz in his Monthly Progress Report for February 1972. Chemical analyses of all head samples, obtained by Grauerholz, are tabulated in Table I.

Split portions of all samples were further prepared by ultrasonic scrubbing methods to separate the slime fraction (approximately minus 450 mesh) and obtain

AUTHOR C. S. Thompson | DATE 3-8-72 | WITNESSED BY R. Labosky | DA 3/

washed material for hand magnetic separation and screen analyses to determine liberation size. A flow sheet for this procedure is shown in Figure 1 and the semiquantitative emission spectrographic analyses of the products are presented in Table II. Examination of the above products indicated that liberation of the scheelite ($CaWO_4$) particles was essentially complete in the 65 x 100 mesh size fraction.

From the data presented in Table II, it can be seen that the tungsten is distributed rather unevenly in all size fractions, with marked variations between trench Samples B1 (2345-92-1) and B2 (2345-92-2). There is a definite tendency for the tungsten to concentrate in the non-magnetic portion, but a significant amount remained with the magnetics, even in the fine fractions (100 x 200 M and -200 M). It should be noted that although the ultrasonic slimes contained significant tungsten values, there was no concentration over the other fractions. These data also indicate that all the tungsten could not be present as scheelite due to a deficiency of calcium relative to tungsten in all samples except the deep drill hole core, B102/3 (2345-92-9), which was unoxidized and reacted favorably to normal beneficiation methods (see Monthly Progress Report for February 1972 by N. L. Grauerholz). These factors indicated that the tungsten in the oxidized trench samples was present not only as scheelite ($CaWO_4$), but also as one or more other mineral phases. Subsequent work on these samples has been directed toward the determination of these tungsten-bearing non-scheelite phases.

To date, two mineral separation methods have been used in attempting to concentrate the scheelite and other tungsten-bearing phases, i.e., magnetic and heavy media separations. For both studies, all samples were ground to minus 65 mesh for liberation and screened into suitable size fractions. The flow sheet for the Frantz magnetic separation is presented in Figure 2 and the emission spectrographic analyses of the products from Sample B1 are tabulated in Table III. The results obtained on this sample were inconclusive since the tungsten was distributed in almost all products, including the hand magnetic fractions.

Trench Samples B1 and B6 were selected for separation by heavy media techniques as shown by the flow sheet in Figure 3. The emission spectrographic data for this test are presented in Table IV. These data show a marked concentration of tungsten in the sink at 3.3 g./ml. (methyl iodide) as would be expected, but in the products from Sample B1 significant quantities remained in the two lighter fractions, both the float and sink at 2.96 g./ml. (acetylene tetrabromide). The differences between Samples B1 and B6 are due mainly to the difference in the degree of oxidation. Sample B6 appears to be only slightly altered relative to B1 with the tungsten present primarily as scheelite. Since Sample B6 was collected from Trench 13, the lowest outcrop topographically in the present series, it is possible that the erosion responsible for the present valley is quite recent and that less oxidized material will be encountered as exploration proceeds to the north.

AUTHOR: C S Thompson | DATE: 3-8-72 | WITNESSED BY: [illegible] | DAT: 3/

Mineralogical studies by X-ray diffraction and microscopic techniques are being conducted on all products obtained during the various tests, but to date, with one exception, no tungsten-bearing mineral other than scheelite has been identified. The one exception was the identification of stolzite ($PbWO_4$) in a float concentrate prepared by N. L. Grauerholz from trench Sample B4. In this concentrate, which assayed ~19% WO_3 but contained only 6% of the total tungsten, stolzite was the major tungsten mineral with lesser quantities of scheelite.

At present, all other samples have been prepared for heavy media and magnetic separations by further grinding to -100 mesh size to ensure liberation of the tungsten-bearing phases. Detailed X-ray diffraction and microscopic examination of all products is in progress to discover the nature and distribution of the tungsten minerals.

TABLE I

Wet Chemical Analyses
of Tasminex Kara No. 1 Head Samples - OR 579

Niagara Falls No.	Field No.	% WO_3	% SiO_2	% Fe	% Ca	% Al	% Mg
2345-92-1	B1	1.55	4.50	59.02	0.02	2.20	0.05
2	B2	1.37	6.00	52.72	0.01	3.64	0.05
3	B3	1.61	4.80	58.84	0.12	2.20	0.04
4	B4	1.29	6.84	53.65	0.04	3.92	0.07
5	B5	1.81	4.80	58.75	0.01	2.26	0.06
6	B6	0.68	7.46	60.76	0.96	0.96	0.59
7	B102/1	0.96	8.22	54.62	0.02	3.80	0.05
8	B102/2	1.12	6.32	59.42	0.30	1.96	0.09
9	B102/3	0.46	14.48	52.16	4.07	1.03	0.92

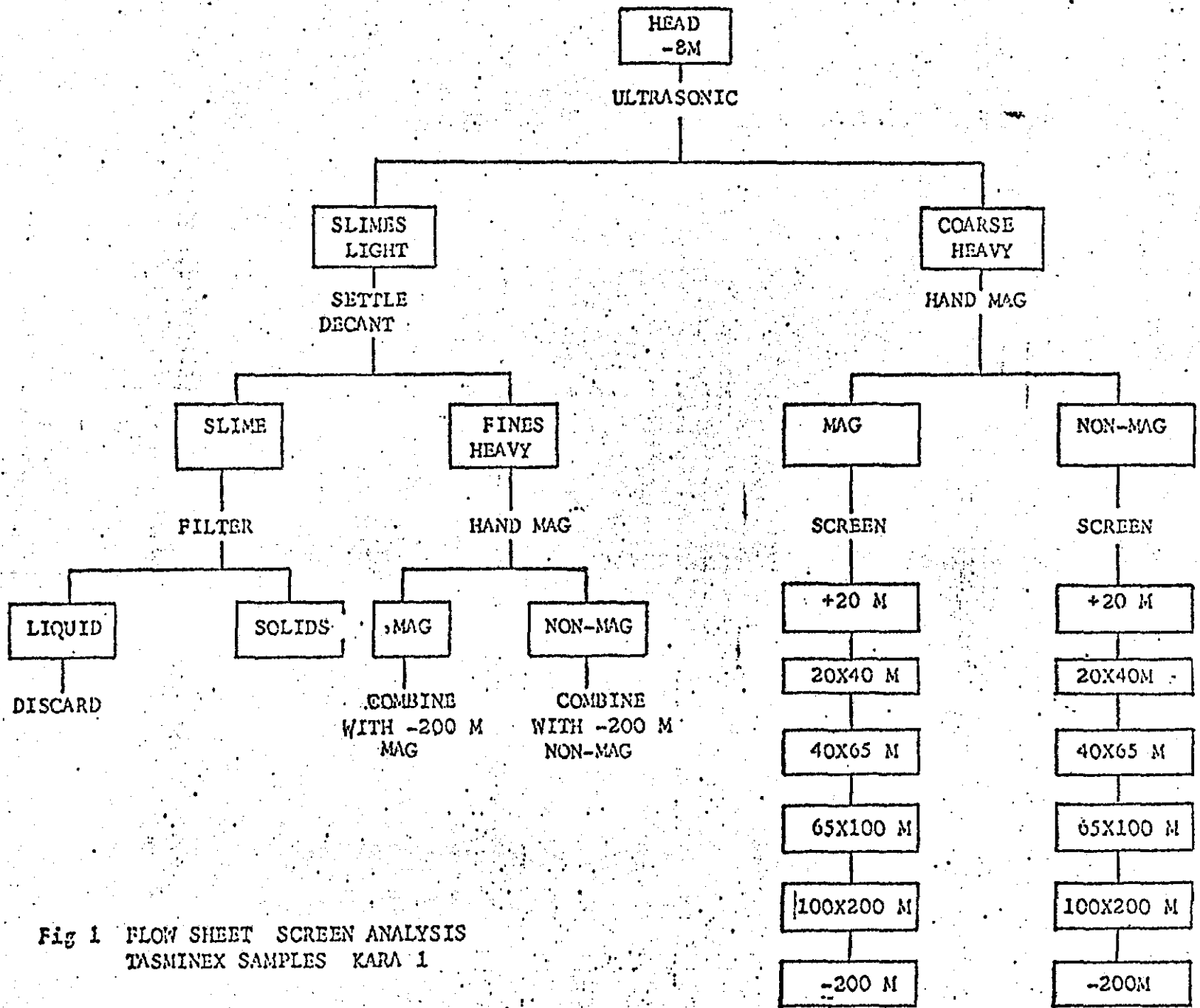

Fig 1 FLOW SHEET SCREEN ANALYSIS
TASMINEX SAMPLES KARA 1

TABLE II

Semiquantitative Emission Spectrographic Analyses of Tasminex Samples OR 579*

Niagara Falls Notebook No.	Field No.	Fraction	Ag	Al	B	Ba	Be	Bi	Ca	Co	Cr	Cu	Fe
345-92-1	B1	Head	-	2.4	-	-	.0024	-	.024	.024	.006	.24	M
1-1-1	B1	+20M Non-Mag.	-	6.	-	-	.0024	.012	.012	-	.006	.24	M
1-1-2	B1	20x40M "	-	6.	-	-	.0024	.012	.03	-	.006	.24	M
1-1-3	B1	40x65M "	-	2.4	-	-	.006	.012	.024	-	.006	.24	M
1-1-4	B1	65x100M "	-	2.4	-	-	.003	.012	.03	-	.003	.24	M
1-1-5	B1	100x200M "	-	2.4	-	-	.003	.024	.03	-	.003	.24	M
1-1-6	B1	-200M "	-	2.4	-	-	.003	.024	.03	-	.003	.24	M
1-2-1	B1	+20M Hand Mag.	-	.8	-	-	.0024	.006	.006	-	-	.12	M
1-2-2	B1	20x40M "	-	.6	-	-	.0024	.006	.006	-	-	.09	M
1-2-3	B1	40x65M "	-	.6	-	-	.0024	.009	.006	-	-	.06	M
1-2-4	B1	65x100M "	-	.6	-	-	.0024	.009	.006	-	-	.06	M
1-2-5	B1	100x200M "	-	.6	-	-	.0024	.006	.006	-	-	.09	M
1-2-6	B1	-200M "	-	.9	-	-	.0024	.006	.006	-	-	.09	M
1-3	B1	Ultra-sonic Slimes	-	3.	-	-	.0024	.012	.012	-	.0024	.12	M
92-2	B2	Head	-	2.4	-	-	.0024	.024	.006	-	.006	.24	M
2-1-1	B2	+20M Non-Mag.	-	2.4	-	-	.0024	.012	.006	-	.003	.24	M
2-1-2	B2	20x40M "	-	2.4	-	-	.0024	.012	.006	-	.003	.24	M
2-1-3	B2	40x65M "	-	2.4	-	-	.0024	.024	.009	-	.0024	.24	M

Woolery

662167

MINERALOGY AND RELATED STUDIES OF TASMINEX

DRILL HOLE SAMPLES RECEIVED IN JUNE 1972

I Split 25-50 grams from each drill hole sample for X-ray diffraction and semi-quantitative spectrographic analysis. These snalyses will be used to determine the abundance and distribution of the major mineral and chemical constituents of the samples recieved. This will form a framework for the coordination of detailed studies of composite and selected samples and for any geologic concepts concerning methods of emplacement and alteration of the deposit.

II Obtain 5-10 pound splits of composites A, B and C from "coarse" 1/2 split.

A. Briquett and polish portion from each composite for optical examination.
   1. Identify phases, describe intergrowths and interpret petrology to extent possible with pre-crushed samples.
   2. Estimate liberation size.

B. Make various separations (such as magnetic, density, hand picking) for detailed mineralogic characterization of each category. Identify constituents of fractions by X-ray, optical and chemical techniques.

III Support of beneficiation tests.
Provide mineral determinations of products from various beneficiation stages such and magnetic, gravity, flotation and roasting.

IV Semi-detailed mineralogic-petrologic and geochemical study of selected individual drill hole samples to determine physical, chemical and geologic processes that have effected ore emplacement, alteration and distribution of primary (unoxidized) and secondary (oxidized) ore types.

File - Tungsten - Tasminex (new)

662168

INTERNAL CORRESPONDENCE

MINING AND METALS DIVISION • P. O. BOX 579, NIAGARA FALLS, NEW YORK 14302

To (Name) Mr. L. A. Wright
Division Mining & Metals Division
Location Union Carbide Corporation
270 Park Ave.
New York, New York 10017

Date January 14, 1972

Originating Dept. Research and Development Department

Answering letter date

Copy to Messrs. D. J. Hansen
A. W. Heuck
D. L. Mathias
J. C. Stephenson
L. P. Twichell
R & D File

Subject Amenability, Mineralogy Tests
Tasminex Samples

We have completed our laboratory examination of the two Tasminex hand specimen samples of your letter of December 1, 1971. These samples are identified as follows:

Sample 1 - Tasminex, Kara 1 Project DDH102, 58 to 58.5 ft. Unoxidized magnetite, 1/4 of 6" piece drill core. Wt. = 281 grams.

Sample 2 - Tasminex, Kara 1 Project. Trench 3 footage, location unknown. Oxidized surface material. Selected grab samples. Wt. = 95 grams.

The unoxidized sample (Sample 1) responded quite well to wet magnetic separation techniques in that 70% of the tungsten reported to a concentrate assaying 60% WO_3. The bulk of the remaining WO_3 appears to be associated with diopside and reports to a product assaying about 14% WO_3. Recovery of this portion of the values in a high-grade product would probably require alternate means of beneficiation. Tabling is questionable due to the small gravity difference between scheelite and diopside and complicated by the fact that all this material is in the minus 100 mesh fraction. Flotation of scheelite from diopside should be "relatively" easy because the ore appears to be very low in calcite and fluorite.

The oxidized Sample 2 represents a more difficult situation. A wet magnetic separation is the obvious first step but unlike the first sample the grade of concentrate is much lower due to the dilution of the oxidized magnetite. Again the mineral diopside is present but also considerable iron oxides and clays are reporting to the nonmagnetic concentrates. The coarse size may table satisfactorily, assuming the iron oxides have a tendency to slime, but I doubt that a really high-grade concentrate with good recovery would be possible. I would suggest flotation for this material also as a final concentrating step; however, I would suspect that overall WO_3 grade and recovery would be somewhat lower than for the unoxidized ore.

In summary, I would say that the unoxidized ore could be upgraded to a high-grade concentrate with good WO_3 recoveries with only little effort. The oxidized ore could be upgraded to a moderate to high-grade concentrate with some applied

effort but I would guess that WO_3 may be in the 50% range depending on concentrate grade required and the amount of nonscheelite-tungsten present.

We should be more specific when the larger, more representative samples are examined. Further, I would remind you that our analyses to date have been on extremely high grade, hand specimens and certainly are not truly representative of the ore body.

Very truly yours,

R. G. Woolery per off.

R. G. Woolery

RGW/bsn

Research & Development Department
P.O.Box 579, Niagara Falls, N. Y.
January 10, 1972

M E M O R A N D U M

TO: Mr. R. G. Woolery

COPY: Messrs. D. J. Hansen
C. S. Thompson
R & D File

FROM: N. L. Grauerholz
C. S. Thompson

SUBJECT: Amenability-Mineralogic Examination of Tungsten Ore from the Tasminex Project

The handpicked specimens described below were received from Union Carbide Exploration Corporation (see letter to R. G. Woolery from Lawson Wright December 1, 1971):

Sample No. 1 - Tasminex, Kara 1 Project. DDH102, 58 to 58.5 ft. Unoxidized magnetite 1/4 of 6-in. piece drill core. Wgt. 241 grams.

Sample No. 2 - Tasminex, Kara 1 Project. Trench 3 footage location unknown. Oxidized surface material. Selected grab samples. Wgt. 95 grams.

Insufficient material was available for comprehensive amenability testing. Test work was limited to rejection of the magnetic fraction by magnetic separation techniques. Other separation methods should be tried, especially on ores as represented by the oxidized sample. The iron in the oxidized sample has been converted to a less magnetic form and may be more amenable to treatment by other methods such as gravity separation or flotation.

Test Procedure

The two samples were prepared for amenability testing by hand-crushing through 20 mesh (Tyler series). The crushed samples were then mixed and riffle split to obtain a head sample for WO_3 analysis and mineralogical examination. The minus 20 mesh head sample was screened at 100 mesh. The plus 100-mesh fraction was subjected to hand magnetic separation, and the minus 100-mesh fraction was treated by the Davis tube.

The plus 100-mesh fraction was subjected to the field of a bar magnet to remove the strongly magnetic material. The nonmagnetic portion was then treated with a more powerful hand magnet to produce a second magnetic concentrate.

The minus 100-mesh head ore was treated with the Davis tube set at 2.5 amps. The test results for the two samples are shown respectively in Table I and Table II, attached.

A spectrographic analysis of each test product from the two samples is shown, respectively, on Table III and Table IV. These estimations did not reveal any other significant amounts of economic elements except tungsten. Arsenic, bismuth, antimony, copper, manganese, molybdenum, and tin as common detrimental impurities appear to be within acceptable limits. The amount of aluminum in the surface sample was significantly greater than in the unoxidized sample. In other respects the surface ore contained similar quantities of elements as those in the unaltered drill core sample. The biggest difference in the two samples is the highly oxidized condition of the surface sample.

Oxidation as evidenced by the surface sample has altered crystal structure and physical properties to such an extent that simple concentration techniques will not be applicable to this ore. As shown in the attached Table II, both grade and recovery of tungsten values by magnetic concentration alone are poor. A combination of magnetic separation and other concentrating steps such as gravity and flotation will be needed to produce a high grade concentrate at an acceptable recovery.

The unoxidized sample (drill core) was much more amenable to magnetic concentration. However, the minus 100-mesh material (Davis tube nonmagnetic) produced only a 13% WO_3 concentrate. This indicates that the nonmagnetic content of the ore is sufficiently high to dilute the tungsten content and produce a low grade concentrate. Other concentrating techniques in conjunction with magnetic concentration will be necessary to raise the grade of this product.

The WO_3 content of the submitted samples, if representative, indicates that both the oxidized and unoxidized portions of the ore body contain ore grade material. Because of the physical differences in the two submitted samples, a beneficiation process must be flexible. The absence of appreciable calcite in the submitted samples and the small grain size of some of the tungsten values indicates that this ore may be more amenable to flotation than to other processes.

The submitted samples are much too small for amenability tests. For meaningful amenability results, the samples should be sufficiently large to be representative. It should be representative of the ore body in respect to ore grade, impurities, degree of oxidation, mineral grain size, mineral association, and other factors that could affect concentration. Outside factors such as the use of detergents in drilling, artificial drying or contamination with oil and grease can be detrimental to the processing of the ore and should be avoided. A minimum of five pounds of sample is needed for amenability tests. More material is desirable, as alternate test procedures can then be tried.

Mineralogy

Mineralogic examination of the head and each test product of both the drill core (2345-91-1) and oxidized trench (2345-91-2) samples was conducted using X-ray diffraction methods. An estimate of the relative amounts of the minerals present in each sample is presented in Table V. The mineral content of the

unoxidized drill core was rather straightforward with the tungsten being present as scheelite ($CaWO_4$) which was easily separated from the gangue minerals. The scheelite content of the "2nd magnetics" (2345-91-1-2) could be recovered by finer grinding, as liberation appears to be complete in the minus 100-mesh fractions. The mineralogy of the oxidized trench material is somewhat more complex in that although some scheelite remains, a significant portion of the tungsten appears to be present in an amorphous (to X-ray) form associated with the slime fraction. In addition, the magnetite has partially oxidized to hematite and goethite and the mica-like, layered silicate phase present in the core sample has altered to a montmorillonite-type clay material. More extensive study of future samples will be necessary to more closely define the nature and composition of the fine fraction of the oxidized surface material.

N. L. Grauerholz
N. L. Grauerholz

C. S. Thompson RL
C. S. Thompson

NLG/CST:rvg

TABLE I

Magnetic Concentration of Tasminex, Kara 1 Project, Ore

Sample No. 1: Tasminex, Kara 1 Project, DDH102, 58.0-58.5 ft. Unoxidized Magnetite

Sample	Product	% Weight	% WO_3	% WO_3 Distribution
	+100 Mesh			
2345-91-1-3	Nonmagnetic	6.0	61.34	86.1
2345-91-1-2	2nd Magnetic	12.3	3.66	10.6
2345-91-1-1	1st Magnetic	81.7	0.17	3.3
	Calc. Feed	100.0	4.25	100.0
	-100 Mesh			
2345-91-1-5	Davis Tube Nonmag.	34.8	13.88	98.9
2345-91-1-4	Davis Tube Mag.	65.2	0.08	1.1
	Calc. Feed	100.0	4.88	100.0
	Combined + and -100 Mesh			
2345-91-1-3	+100 Nonmag.	4.9	61.34	69.3
2345-91-1-5	Davis Nonmag.	6.1	13.88	19.3
2345-91-1-2	+100 Mag. No. 2	10.2	3.66	8.6
2345-91-1-1	+100 Mag. No. 1	67.4	0.17	2.6
2345-91-1-4	Davis Tube Mag.	11.4	0.08	0.2
	Calc. Feed	100.0	4.36	100.0
	Assay Feed		3.71	
	Combined Total Ore			
	Total Nonmag.	11.0	35.15	88.6
	Total Mag.	89.0	0.56	11.4
	Calc. Feed	100.0	4.36	100.0

REF. 1832-28

TABLE II

Magnetic Concentration of Tasminex, Kara 1 Project, Ore

Sample No. 2: Tasminex, Kara 1 Project, Trench 3, Footage Location Unknown, Oxidized Surface Material

Sample	Product	% Weight	% WO_3	% WO_3 Distribution
		+100 Mesh		
2345-91-2-3	Nonmagnetic	9.3(6.9)	25.42	44.4
2345-91-2-2	2nd Magnetic	6.3(4.6)	10.18	12.1
2345-91-2-1	1st Magnetic	84.4(62.8)	2.74	43.5
	Calc. Feed	100.0(74.3)[1]	5.32	100.0
		-100 Mesh		
2345-91-2-4	Davis Tube Nonmag.	61.5(15.8)	10.28	91.1
2345-91-2-5	Davis Tube Mag.	38.5(9.9)	1.61	8.9
	Calc. Feed	100.0(25.7)[1]	6.94	100.0
		Combined + and -100 Mesh		
2345-91-2-3	+100 Nonmag.	6.9	25.42	30.6
2345-91-2-4	Davis Tube Nonmag.	15.8	10.28	28.3
2345-91-2-2	+100 Mag. No. 2	4.6	10.18	8.3
2345-91-2-1	+100 Mag. No. 1	62.8	2.74	30.0
2345-91-2-5	Davis Tube Mag.	9.9	1.61	2.8
	Calc. Feed	100.0	5.73	100.0
	Assay Feed		4.48	
		Combined Total Ore		
	Total Conc.	27.3	14.09[2]	67.2
	Total Tail	72.7	1.88	32.8
	Calc. Feed	100.0	5.73	100.0

[1]Distribution by total ore.
[2]Includes weaker magnetic material 2345-91-2-2.

TABLE III

Report of Spectrographic Analysis
Tasminex, Kara 1 Project, Samples

Sample No. 1: Values in % on Sample Basis

Sample Product	1832-28-1 Head	2345-91-1-3 +100 Nonmag.	2345-91-1-5 Davis Tube Nonmag.	2345-91-1-2 +100 Mag. No. 2	2345-91-1-1 +100 Mag. No. 1	2345-91-1-4 Davis Tube Mag.
Ag	.0004-.004	-	.001-.01	.001-.01	-	-
Al	.08-.8	.08-.8	.2-2	.2-2	.08-.8	.04-.4
Be	.0004-.004	.0008-.008	.0008-.008	.0008-.008	.0004-.004	.0004-.004
Bi	.008-.08	.02-.2	.03-.3	.02-.2	.004-.04	-
Ca	.3-3	M	1-10	2-20	.04-.4	.008-.08
Co	.01-.1	-	-	-	.008-.08	.008-.08
Cr	-	-	.002-.02	-	-	-
Cu	.08-.8	.03-.3	.1-1	.2-2	.03-.3	.03-.3
Fe	M	.3-3	M	M	M	M
Ga	.004-.04	-	-	-	-	-
Mg	.2-2	.8-8	.8-8	.8-8	.08-.8	.08-.8
Mn	.08-.8	.02-.2	.03-.3	.04-.4	.08-.8	.08-.8
Mo	.02-.2	.2-2	.08-.8	.01-.1	.003-.03	.002-.02
Ni	.002-.02	-	-	-	-	.002-.02
P	-	.08-.8	-	-	-	-
Pb	.04-.4	.1-1	.08-.8	.08-.8	.02-.2	.02-.2
Si	2-20	M	M	M	.2-2	.08-.8
Sn	.02-.2	.008-.08	.02-.2	.02-.2	.03-.3	.02-.2
Ti	.008-.08	-	.002-.02	.002-.02	.008-.08	.008-.08
V	.004-.04	.001-.01	-	-	-	-
W	2-20	M	2-20	1-10	.01-.1	-
Zn	.01-.1	-	.01-.1	.01-.1	.01-.1	.01-.1

Looked for but not detected: As, Au, B, Ba, Cb, Cd, Co, Hf, Hg, In, Ir, Li, Na, Ni, Os, P, Pd, Pt, Rh, Sb, Sr, Ta, Th, Tl, U, V, Zr

TABLE IV

Report of Spectrographic Analysis
Tasminex, Kara 1 Project, Samples

Sample No. 2: Values in % on Sample Basis

Sample Product	1832-28-2 Head	2345-91-2-3 +100 Nonmag.	2345-91-2-4 Davis Tube Mag.	2345-91-2-2 +100 Mag. No. 2	2345-91-2-1 +100 Mag. No. 1	2345-91-2-5 Davis Tube Nonmag.
Al	2-20	1-10	.8-8	M	.8-8	M
Be	.0004-.004	.0004-.004	.0004 .004	.0004-.004	.0004-.004	.0004-.004
Bi	.008-.08	.004-.04	.01-.1	.008-.08	.008-.08	.01-.1
Ca	.08-.8	.8-8	.01-.1	.08-.8	.008-.08	.1-1
Cu	.04-.4	.04-.4	.04-.4	.08-.8	.08-.8	.08-.8
Fe	M	2-20	M	2-20	M	2-20
Ga	.004-.04	.08-.8	-	-	.008-.08	-
Mg	.08-.8	.08-.8	.08-.8	.08-.8	.08-.8	.1-1
Mn	.08-.8	.008-.08	.1-1	.008-.08	.1-1	.008-.08
Mo	.03-.3	.1-1	.008-.08	.02-.2	.01-.1	.02-.2
Pb	.02-.2	.02-.2	.02-.2	.03-.3	.03-.3	.04-.4
Si	2-20	M	.8-8	M	2-20	M
Sn	.04-.4	.08-.8	.04-.4	.08-.8	.08-.8	.08-.8
Ti	.03-.3	.03-.3	.02-.2	.03-.3	.02-.2	.04-.4
W	2-20	M	2-20	2-20	2-20	2-20
Zn	.01-.1	-	.02-.2	-	.02-.2	.01-.1
Zr	.003-.03	.002-.02	-	.002-.02	-	.002-.02

Looked for but not detected: Ag, As, Au, B, Ba, Cb, Cd, Co, Hf, Hg, In, Ir, Li, Na, Ni, Os, P, Pd, Pt, Rh, Sb, Sr, Ta, Th, Tl, U, V

TABLE V

X-ray Diffraction Estimations of Mineralogic Content of Tasminex Tungsten Samples and Their Amenability Test Products

Sample	Product	Magnetite	Hematite	Scheelite	Diopside	"Mica"	Goethite
2345-91-1	Head Sample	Major	Minor	Some	Some	Trace	-
1-1	1st Magnetic	Major	Minor	-	-	-	-
1-2	2nd Magnetic	-	Trace	Minor	Major	Minor	-
1-3	Nonmagnetic	-	-	Major	Moderate	-	-
1-4	Davis Tube Mag.	Major	Minor	-	-	-	-
1-5	Davis Tube Nonmag.	-	Trace	Major	Moderate	Minor	-
						"Mont-morillonite"	
2345-91-2	Head Sample	Major	Major	Some	-	Some	-
2-1	1st Magnetic	Major	Major	-	-	Some	-
2-2	2nd Magnetic	Minor	Trace	Trace	Trace	Major	Some
2-3	Nonmagnetic	-	Trace	Major	Trace	Moderate	Some
2-4	Davis Tube Mag.	Major	Major	-	-	-	-
2-5	Davis Tube Nonmag.	-	Trace	Trace	-	Major	Some

Major	> 25
Moderate	15-25
Minor	10-15
Some	5-10
Trace	< 5