RED FLATS NICKEL PROJECT

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I. INTRODUCTION

A. Purpose of Report

This study by the Pacific Nickel Corporation of Los Angeles reports the progress to date in the investigation and treatment of the nickeliferous ores in the Red Flats area of Southwestern Oregon.

The exploration work on these Pacific Nickel Corporation deposits has included field work, aerial surveying, trenching, drilling, sampling, assaying and economic study.

Pacific Nickel Corporation conducted field and laboratory investigations during 1956 and 1957 aimed at the profitable treatment of these ores.

This work led to the possible application of the Krupp-Renn process, which is attracting world wide interest, for the production of nickeliferous iron from similar ores.

B. Location and Access

The deposits of the Pacific Nickel Corporation are located in Section 19 and 30, T. 37 S., R. 13 W., eight air miles, seventeen road miles east of Gold Beach, Curry County, Oregon, in the extreme southwest coroner of the State.

The Oregon coast highway (U.S. 101) follows along the narrow coastal plain connecting the small towns of Brookings, Gold Beach and Port Orford.

Access is by a Forest Service road leading to the Snow Camp Mountain Lookout. This road leaves U.S. Highway 101 at Hunters Creek, three miles south of Gold Beach.

C. History

These deposits were located as gold-mercury placer prospects in the early thirties. Small mills and retorts were set up on the property at different times. A few shallow hand trenches and a thirty-two foot shaft comprised the early development work. In 1945 and 1946, four or five shallow bulldozer trenches were excavated, and three or four small camp buildings were constructed by the operators. These small operations were not successful and most of the equipment was removed.

The property was relatively dormant from 1947 to 1953, when the Bureau of Mines subjected a fifteen-ton sampling of the material to continuous smelting tests in the Bureau's Northwest Electrodevelopment Laboratory, Albany, Oregon.

Neither the field investigations nor the smelting tests were considered complete.

The details of this testing are very adequately covered in Bureau of Mines Report of Investigations 5072, "Preliminary Investigation of the Red Flats Nickel Deposits, Curry County, Oregon."

An extremely thorough study has been made of the Coos Bay coal deposits entitled "Geology and Coal Resources, Coos Bay Quadrangle, Oregon", by Allen and Bladwin, known as Bulletin Number 27 issued by the State of Oregon Department of Geology and Mineral Industries in 1944.

II. SUMMARY AND CONCLUSIONS

The Red Flat Nickel Project has been investigated in a preliminary way by the Pacific Nickel Corporation of Los Angeles, California. Exploration to date on about 5 percent of the property indicates an ore body in excess of 3,000,000 tons of measured ore on this 5 percent portion.

If further work bears out preliminary assumptions, the ore will be amenable to the Krupp-Renn process.

This process will treat the ore to produce a material which will run about 4.2% nickel and 90% iron. Such a product in the present day nickel shortage may be salable at prices of 74¢ and up per pound of contained nickel. No market analysis has been made. However, similar material in Europe has been sold during the current year.

If these assumptions are correct, a profit in excess of $7,000,000 per year might be expected on an investment of less than $17,000,000.

This progress report was intended only as a guide in making a decision to do further work. It is not intended as an offer to do the work at the mentioned prices or a guarantee of the process.

Additional analyses of both the deposit, the metallurgy and the market must be made before a decision to build the plant can be reached.

III. DESCRIPTION OF DEPOSIT

A. Physical Features and Climate[1]

The Red Flats deposit is on a relatively flat-topped ridge between Hunters Creek on the west and Pistol River on the east. The lower slopes of the ridge are deeply dissected. The altitude of the deposit ranges between 2,150 and 2,500 feet above sea level.

Large areas of the deposit are relatively barren of timber and covered only by dense brush. Scattered patches of knob cone pine, stunted oaks, and manzanita are elsewhere on the deposit but have little value for mining purposes. West of the deposit, timber of excellent quality is found in large quantity.

Typical Oregon coastal climate prevails with heavy rainfall, fog, and storms common during the winter and spring months. Snow and freezing temperatures are exceptional. Annual precipitation averages about 50 inches. The summers are dry and mostly clear, and there is little temperature variation, the yearly mean average temperature being around 50° F.

Water is available from Pistol River or Hunters Creek at all times. The higher slopes of the ridge are comparatively dry. One permanent spring near the summit of the ridge supplies enough water for domestic and limited industrial use.

1 Quoted from page 3 Bureau of Mines Report of Investigations 5072, Preliminary Investigation of the Red Flats Nickel Deposit, Curry County, Oregon, by R. J. Hundhausen, J. R. McWilliams, and L. H. Banning.

B. Mineralogy

Report on Petrography and Mineralogy of Serpentine and Laterite for Southwestern Engineering Company March 14, 1957

I. Serpentine

Minerals:

Antigorite-80-90%. Colorless to pale greenish, moderate to low + relief, birefringence 1st order gray to white in thin section. Index of refraction 1.561. Occurs as intergrown plates and lamellae composing the bulk of the rock. Some along tiny veinlets is finely intergrown with iron oxide (mainly limonite) and has a slightly higher birefringence than the "islands" of colorless serpentine it surrounds. The material of the islands corresponds in appearance to the variety of serpentine mineral which has been called serpophite. The islands of serpophite are 0.2-0.3 mm. in diameter, while the individual crystals of serpophite composing them, as well as the antigorite crystals surrounding them, are up to about 0.05 mm. in diameter.

Chrysotile- 3%. Colorless, moderate + relief, birefringence 1st order white to yellow. Occurs as veinlets of fibrous individuals oriented perpendicular to the veinlet walls. Iron oxide is commonly present in varying quantities up to about 50% in these veinlets, which are up to 0.5 mm. wide.

Limonite- 2-5%. Red in transmitted and reflected light, very fine grained, apparently isotropic. Intergrown with the serpentine minerals as mentioned above, and also as patches up to 0.1 mm. diameter with the islands of serpophite.

Hematitie- 2-5%. Red in transmitted and reflected light, but much deeper color than the limonite, anisotropic, high relief. As grains 0.1 mm. and less scattered through the whole rock, and as concentrations along small fractures.

Chromite- 1%. Opaque, metallic but dull luster, dark brown on thin edges. 0.02-0.1 mm. diameter. Frequently as euhedral grains with octahedral shape. Very slightly magnetic.

Magnetite- 2-5%. Opaque, metallic luster, strongly magnetic, sometimes euhedral, but mostly as irregular grains from 0.01-0.1 mm. diameter.

Pyroxene? - trace. High relief, greenish, high birefringence. Two small remnants noted. Might also be olivine.

Pyrite- 1/2%. Brassy yellow color, cubic shape. 0.05 mm. in size.

Opal? - trace. Greenish, almost isotropic, strong-relief. Occurs as a 0.5 mm. veinlet with magnetite.

The rock is a typical serpentine, probably developed by alteration of an olivine, or pyroxene-rich rock during the process of cooling. The chromite and part of the magnetite probably represent original constituents of the peridotite. The limonite, hematite, and some of the magnetite were apparently developed during alteration of the serpentine by waters percolating thru the rock from the surface. Limonite appears to be most abundant in the initial stages of the latter alteration, with more and more hematite later developing along cracks.

The foregoing description applies to two thin sections out from the freshest chunks of rock in the sample, plus examination of crushed fragments of more weathered chips. The material in the "representative rock sample" differs from the above description by the presence of appreciably greater alteration to iron oxides in many of the fragments, and along with this a decrease in the amount of serpentine minerals.

II. Laterite

Minerals:

Hematite - 60-90%. Dark red to opaque. Grain size 0.22 mm. and less.

Limonite - 5-25%. Yellowish red. Grain size similar to hematite.

Magnetite - 5-10%. Opaque, metallic luster, strongly magnetic. 0.01-0.1 mm.

Chromite - 1-2%. Opaque, dull metallic luster, weakly magnetic. 0.02-0.1 mm.

Serpentine - 2-5%. Low birefringence, moderate + relief, finely intergrown with limonite.

An x-ray powder pattern of the laterite was made in an attempt to confirm the minerals believed to be present, and find other minerals which might have been missed owing to the generally fine-grained and opaque character of the material. The following lines were recorded:

d spacing	minerals
4.25 A	limonite?
3.68	hematite
3.35	?
2.70	hematite
2.51	hematite
2.20	hematite
1.69	hematite

The peaks were broad and of low intensity, suggesting that the material is very fine grained or else poorly crystalline.

The material is composed dominantly of hematite, developed apparently by a continuation of the alteration and weathering process observed in the serpentine. The chromite and much of the magnetite, as well as the small amounts of serpentine minerals, appear to represent remnants of the original serpentine rock. A considerable amount of fine-grained magnetitie seems to have developed in the weathering process, and of course hematite has become the dominant mineral.

/S/ Arthur W. Rose

C. Geology [2]

The general geology of this region is described by Butler and Mitchell.[3]

The nickeliferous laterite rests on a weathered bedrock of ultrabasic rocks, principally peridotite and serpentinized peridotite. These rocks are part of the Josephine peridotite intrusives common in southwestern Oregon and northern California. The nickeliferous laterite is confined to the surface of the peridotite and has been derived from it under certain conditions of weathering, particularly where there are alternating cycles of hot dry climate followed by a prolonged rainy season. Where the peridotite contacts Colebrooke schists (pre-Jurassic) and the Dothan formation (Jurassic) immediately south

2 IDID - Reference 1, Page 4
3 Butler, G. M., and Mitchell, G. J., Preliminary Survey of the Geology and Mineral Resources of Curry County, Oregon: Oregon Bureau of Mines and Geology, Mineral Resources of Oregon, Vol.2, No.2, October 1916, 136pp.
4 Wells, F. G., Hotz, P. E. and Cater, F. W., Jr., Preliminary Description of the Geology of the Kerby Quadrangle, Oregon: Oregon State Department of Geology and Mineral Industries Bull. 40, 1949, 23pp.

of the Red Flats area, the laterite is absent.

The comparatively low iron content and the high magnesium and silicon contents of the lat erite in the Red Flats area indicate a relatively youthful stage of weathering. If more mature laterites, high in iron content, were ever developed in this area, they have since been eroded, and there is nothing to indicate their former presence. The iron content of the laterite ranges in grade from 25 to 45 percent. Residual boulders of undecomposed peridotite are scattered throughout the laterite. The laterite has accumulated along the axis of the ridge and in small buasins on the east slopes sheltered or rimmed by resistant dikes or ledges. The west slope of the ridge is covered with only a thin veneer of red soil (laterite).

Nickeliferous Laterite Areas

The nickel-rich laterite areas have not been completely delimited by the Bureau of Mines drilling program. Apparently more of this laterite is present north of the area explored. The nickeliferous laterite areas are outlined in figure 2. A cutoff grade of 0.90 percent combined nickel and cobalt was used to establish the boundaries of higher grade laterite areas. The depth of nickel-rich laterite and the grade are shown for each hole. None of the areas contain overburden except hole 17, in which 5 feet of relatively barren laterite is on top of the nickel-rich laterite.

The shape and form of the nickel-rich laterite areas are variab le; the nickel content of the laterite is variable, and some of the material is a mixture of residual laterite and transported laterite.

The nickel content of the laterite cannot be estimated by visual examination but must be determined by analytical methods. The nickel content does not necessarily depend on the degree of weathering or the depth of laterite soil. A few surface indications have been recognized that roughly aid in prospecting for nickel-rich laterites, but these are not infallible guides. Surface accumulations of rounded, shot-sized pellets of iron oxides in the soil are fair indicators of a relatively higher nickel content in the laterite. Furthermore, a loosely consolidated, porous, soft, well-drained, residual type laterite generally contains more nickel than a clayey, dense, compact, transported type laterite. Residual laterite usually has a higher specific gravity than a clayey type or transported laterite.

Chemical and Mineralogical Composition of Laterite

No discrete nickel minerals were observed in the laterite. The nickel is believed to be in a finely divided form as hydrated nickel oxides; some is chemically combined in the limonite mineral structure; a little may be with magnetite and chromite. It is not possible to improve the grade of nickeliferous laterite by handsorting nor has beneficiation by ore dressing been successful to date.

The chemical composition of nickeliferous laterites is surprisingly uniform throughout the world. These laterites are only formed on or derived from serpentinized ultrabasic rocks, which apparently are relatively uniform in composition. The nickeliferous laterites are quite different in composition from laterites derived from the more highly differentiated silicic rocks.

Serpentine rocks generally lack vegetation. A serpentine rock must undergo a certain amount of chemical decomposition before mechanical disintegration and laterite formation. The ultramafic silicate minerals, such as olivine, have iron, nickel, chromium, aluminum, calcium, and magnesium tied up in their chemical structures. These minerals lack chemical stab ility under surface weathering, oxidizing conditions, but they do not disintegrate immediately. They hydrolyze and become partly hydrated and recrystallize, forming more stable hydrous silicates. This is a gradual rock-softening process, with very little change in net composition. As this reaction proceeds, the soluble bases and silicic acid are more easily removed by solution in ground water, and more complete chemical decomposition follows. Colloidal-size particles then may re removed rapidly by mechanical means, together with the chemical solution of the soluble bases. Under favorable topographic and climate conditions, the insoluble bases and heavy minerals accumulate to form residual laterite deposits.

For the reasons mentioned above, the contact between the nickeliferous laterite and the underlying bedrock is sharp, although extremely irregular. There are abrupt changes in color, in handness, in physical appearance, and most important of all, in chemical composition.

Table 1 gives the partial analyses of some representative samples of nickeliferous laterites and nickeliferous serpentine deposits throughout the world. Analyses of the Red Flats material are shown also for purpose of comparison.

Table 1 - Partial analyses of representative types of nickel ores.

	Nickeliferous laterites				Nickeliferous serpentines				
	1	2	3	4	5	6	7	8	9
Fe	46.81	52.08	49.36	38.5	10.2	18.01	9.7	12.21	11.52
Cr	2.05	2.04	1.68	2.12	.77	.91	.74	.66	.73
Ni	.72	1.02	.30-.64	.92	1.29	1.08	1.50	3.48	6.17
Co	.13	.08	unavail-able	.29	.17	.08	unavail-able	.11	.12
SiO_2	1.62	2.76	2.04	7.58	37.5	41.94	45.8	41.55	41.4
Al_2O_3	9.41	6.61	5.04	10.76	2.0	3.48	3.4	.16	.92
CaO	-	.15	.74	-	1.1	.25	1.0	-	.08
MgO	.80	.72	.07	5.47	17.2	12.09	19.9	22.97	22.8

Note. - Column heads: (1) Oriente Province, Cuba - average nickel laterite. Assays by Bureau of Mines; (2) Oriente Province, Cuba - medium-grade nickel laterite. Assays by Bureau of Mines; (3) Celebes iron ore - lake district near Larona.; (4) Red Flats laterite. Assays by Bureau of Mines; (5) Red Flats Nickeliferous altered serpentine. Assays by Bureau of Mines. (6) altered serpentine - Oriente Province, Cuba. Assays by Bureau of Mines; (7) Riddle nickel deposit, Riddle, Oregon - 310-ton sample assayed by Bureau of Mines; (8) sorted ore - Celebes; (9) sorted ore - New Caledonia.

From the analyses shown above, the nickeliferous laterites are enriched in iron, alumnia, chromium, and probably cobalt contents, compared to the nickeliferous serpentines. The magnesia and silica contents in the laterites are heavily depleted in comparison to the serpentines. These relationships generally are valid, no matter how the nickel content varies in the laterite. The nickel content of most laterites is higher than that of most of the serpentine on which it forms. The amount of nickel that can be accumulated in the soil apparently is limited by its solubility. After a certain maximum degree of residual enrichment is obtained, the nickel no longer accumulates but instead is leached out of the soil, judging from the fact that many of the enormous nickel laterites of the world have uniform nickel contents averaging close to 0.6 to 0.7 percent nickel. Assuming that most serpentines average close to 0.10 percent nickel over large areas, the ratio of concentration of nickel in the soil due to weathering is generally about 6 or 7 to 1. Some relatively small areas of serpentine average 0.3 to 0.5 percent nickel. When these areas are laterized, the nickel content in the resulting residual laterite may accumulate to an average maximum of 1.8 to 3.5 percent nickel, calculated at the same ratio of concentration as before. These richer areas of laterite constitute the minable deposits in Cuba. Most residual laterites show variations in nickel content that reflect to a marked degree the primary differences in the nickel content of the serpentinized rock that has been weathered. Transported laterite of course would not exhibit this relationship with the underlying bedrock.

Nickeliferous Serpentines

The nickel-rich serpentinized bedrock in the Red Flats area, as determined by assays of the Bureau of Mines drill-hole samples, is outlined on figure 2. A cutoff grade of 1 percent combined nickel and cobalt was used to delimit the boundaries. The depth of overburden on this nickeliferous serpentine is shown for each hole. Overburden may consist of barren laterite, nickel-rich laterite, or relatively barren rock.

Lack of development of this bedrock deposit precludes accurate description. The nickeliferous area may be more irregular than is indicated.

The general trend of the nickeliferous zone is in a northerly direction, coincident with the axis of the ridge. This zone has an explored length of 3,900 feet, as determined by the drilling; the zone may extend farther to the north beyond the area explored. The extension of the zone to the south is limited. The average width is 590 feet and the average thickness is 12.5 feet.

Garnierite (hydrous nickel silicate) is exposed in the serpentinized peridotite in the trench by drill hole 15 near the south end of the zone. This trench is a few hundred feet north of the permanent springs that supply the Red Gold mining camp. The moist soil in the vicinity of the springs supports a luxuriant growth of Darlingtonia (pitcher, cobra, or fly-catcher plants). Garnierite was found at Woodcock Mountain in a similar environment just above a side hill springs area supporting a conspicuous growth of Darlintonia. These plants may be indicators of nickel-rich zones in the serpentinized ultrabasics of southern Oregon.

D. Ore Reserves

1. Definitions

This section is included in the report to clarify the terms used. In the process of evaluating mining properties there are certain terms that can be misleading if not properly qualified. Particular reference is made to the terms used in making estimates of ore tonnages. The U. S. Bureau of Mines and the U. S. Geological Survey have agreed upon the following terms and definitions of these terms to signify relative dependability of field information.

"'Measured ore' is ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are so closely spaced and the geological character is so well defined that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to be accurate within limits which are stated, and no such limit is judged to differ from the computed tonnage or grade by more than 20 percent."

" 'Indicated ore' is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout."

"'Inferred ore' is ore for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Estimates of inferred ore should include a statement of the special limits within which the inferred ore may lie."

The term "ore" is often loosely used to designate anything recovered from the earth by mining processes. Technically "ore" is an aggregation of mineral and waste from which ore or more metals may be extracted at a profit. It is obvious then that the market conditions and the efficiency of the mining process will determine whether or not a mineralized area has any "ore".

In reading the term "ore" in estimates within the report, it must be kept in mind that this material referred to may not be of economic importance due to low grade, impurities, low market, etc., and that a strict use of the word "ore" is impossible until extensive drilling and engineering studies have been carried out.

2. Ore Reserves

Red Flats Area:	a.	Measured ore	3,000,000 ± tons
	b.	Indicated ore	3,000,000 ± tons
	c.	Inferred ore	10,000,000 ± tons

Grade of Ore:	a.	Average assays	0.91% Ni.
	b.	Typical Analyses	

	Laterite Lab #52050 Sample 3821 Heads		Serpentine Lab #52051 Sample 3822 Heads	
Surface Moisture	22.49)	25.75	7.63)	11.05
	3.26)		3.42)	
Combined Moisture	8.16		14.45	
Ni	.83		.92	
Co	.047		.072	
Total Fe	49.49		8.70	
FeO	2.47		1.60	
Fe_2O_3	65.61		8.89	
Fe_3O_4	5.04		3.37	
SiO_2	6.04		39.08	
Al_2O_3	4.46		.12	
CaO	.09		.11	
MgO	3.05		30.57	
Total Alk. (Na_2O)	.26		.18	
Cr	1.75		.94	
P	.035		.003	
S	.040		.014	
TiO_2	.12		.05	
Cu	.06		.02	
Zn	.06		.04	
Pb	.05		.03	

3. Work by U. S. Bureau of Mines

Drilling

The Red Flats area was drilled on a grid system using a Star churn drill mounted on a 6-wheel-drive truck. Twenty-two holes, 6 inches in diameter and ranging from 20 to 117 feet and averaging 35 feet deep, were drilled in 8 lines 500 feet apart. The holes were drilled in order of their accessibility, and this accounts for the somewhat erratic numbering system. The cutoff point in drilling was generally determined by the increased hardness of the rock, discernible in drilling.

The area was surveyed with a Brunton compass and tape. Elevations were obtained by rod and hand-level method.

Assay Logs of Drill Holes 1 to 23
(Excluding Hole 14)

Assays of Drill Hole No. 1 — Depth 60'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3	% Co
0-5	4-2	0.43	38.3			
5-10	4-3	.40				
10-15	4-4	.31				
15-20	4-5	.88	9.8	14.7	0.62	0.12
20-25	4-6	1.01	7.8			
25-30	4-7	.80	7.2			
30-35	4-8	.48				
35-40	4-9	.37				
40-45	4-10	.29				

Assays of Drill Hole No. 4 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3	% Co
0-5	2-20	1.13	46.0			
5-10	2-21	1.38	38.4	8.96	3.64	0.10
10-15	2-22	.80	14.3			
15-20	2-23	.30				
20-25	2-24	.85				
25-30	2-26	.86				
30-35	3-14	.81				
35-40	3-1	.73				
40-45	3-2	.86				
45-50	3-3	.84	6.4	7.58		

Assays of Drill Hole No. 2 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3
0-5	4-23	0.66			
5-10	4-24	.42			
10-15	4-25	.46			
15-20	5-1	.71			
20-25	5-2	.81			
25-30	5-3	.81			
30-35	5-4	.81			
35-40	5-5	.44			
40-45	5-6	.41			
45-50	5-7	.25			

Assays of Drill Hole No. 5 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3
0-5	6-21	0.19			
5-10	6-22	.67			
10-15	7-1	.46			
15-20	7-2	.62			
20-25	7-3	.66			
25-30	7-4	.66			
30-35	7-5	.39			
35-40	7-6	.24			
40-45	7-7	.88			
45-50	7-8	.21			

Assays of Drill Hole No. 3 — Depth 60'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3	% Co
0-5'	3-15	0.52	26.6			
5-10'	3-16	.61				
10-15'	3-17	.52				
15-20'	3-18	.60				
20-25'	3-19	.90				
25-30'	3-20	.40				
30-35'	3-21	.91	8.8			
35-40'	3-22	1.26	10.6	14.9	0.62	0.04
40-45'	3-23	1.27	10.6			
45-50'	3-24	.80				
50-55'	3-25	.82				
55-60'	4-1	.81	7.6			

Assays of Drill Hole No. 6 — Depth 35'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr_2O_3
0-5	8-25	0.45			
5-10	9-1	.61			
10-15	9-2	.61			
15-20	9-3	.71			
20-25	9-4	.81			
25-30	9-5	.64			
30-35	9-6	.42			

Composite sample of Hole 6 showed 0.06 % Mg

Assays of Drill Hole No. 7 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	2-10	0.81	31.0			
5-10	2-11	1.13	10.8		0.89	0.15
10-15	2-12	1.15	11.8			
15-20	2-13	1.16	10.0	18.2		
20-25	2-14	.81	7.8		.62	.15
25-30	2-15	.49	8.9			
30-35	2-16	.50				
35-40	2-17	.34				
40-45	2-18	1.07	8.4			.12
45-50	2-19	.30				

Assays of Drill Hole No. 9 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	3-4	0.83	31.6			
5-10	3-5	1.30	25.8	15.7	1.66	0.07
10-15	3-6	1.13	6.4			
15-20	3-7	.84	12.2			
20-25	3-8	.43	6.4			
25-30	3-9	.38				
30-35	3-10	.27				
35-40	3-11	.25				
40-45	3-12	.61				
45-50	3-13	.50	17.4			

Assays of Drill Hole No. 10 — Depth 55'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	10-1	0.75	36.6			
5-10	10-2	1.24	35.5			
10-15	2-1	1.13	10.8		0.86	0.08
15-20	2-2	1.20	23.4	8.86		.08
20-25	2-3	.87	18.8			.09
25-30	2-4	.68				
30-35	2-5	.34				
35-40	2-6	.26				
40-45	2-7	.24				
45-50	2-8	.23				
50-55	2-9	.26				

Assays of Drill Hole No. 8 — Depth 70'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	5-18	0.48			
5-10	5-19	.48			
10-15	5-20	.26			
15-20	5-21	.40			
20-25	5-22	.36			
25-30	5-23	.38			
30-35	5-24	.44			
35-40	5-25	.32	8.18		
40-45	6-1	.18			
45-50	6-2	.24			
50-55	6-3	.24			
55-60	6-4	.28			
60-65	6-5	.26			
65-70	6-6	.25			

Assays of Drill Hole No 11 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	9-8	0.33	28.6			
5-10	9-9	.67				
10-15	9-10	.64				
15-20	9-11	.97	19.4			
20-25	9-12	.95	11.0			
25-30	9-13	.84	10.4		0.74	
30-35	9-14	1.25	10.4	13.2		0.13
35-40	9-16	.97	8.8			
40-45	9-16	.59				
45-50	9-17	.58	7.6			

Assays of Drill Hole No. 12 — Depth 40'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr2O3	% Co
0-5	5-8	0.73	21.9			
5-10	5-9	.82	24.4	13.2	1.65	0.13
10-15	5-10	.87	18.8			
15-20	5-11	.46	10.0			
20-25	5-12	.47				
25-30	5-13	.39				
30-35	5-14	.56				
35-40	5-15	.24				

Assays of Drill Hole No. 15 — Depth 117'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr2O3	% Co
0-5	1-1	1.67	43.0		4.2	
5-10	1-2	1.32	30.1		1.5	
10-15	1-3	.67	8.6		.75	
15-20	1-4	.44	8.3		.67	
20-25	1-5	.47	6.0		.46	
25-30	1-6	.40	6.2		.45	
30-35	1-7	.29	5.6		.37	
35-40	1-8	.32	5.75		.46	
40-45	1-9	.31	5.65		.46	
45-50	1-10	.30	4.9		.46	
50-55	1-11	.31	5.36		.46	
55-60	1-12	.28	5.36		.46	
60-65	1-13	.28	5.3		.37	
65-70	1-14	.28				
70-75	1-15	—				
75-80	1-16	—				
80-85	1-17	.26				
85-90	1-18	.26				
90-95	1-19	.28				
95-100	1-20	.38	6.0	1.71	2.50	0.17
100-105	1-21	.57				
105-110	1-22	.20				
110-115	1-23	.23				
115-117	1-24	.24				

Assays of Drill Hole No. 13 — Depth 60'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr2O3	% Co
0-5	6-7	0.71	17.0			
5-10	6-8	.42				
10-15	6-9	.70	7.4			
15-20	6-10	.88	7.6	20.4	0.88	0.02*
20-25	6-11	.89	7.9			
25-30	6-12	1.08	6.4			
30-35	6-13	.61	6.5			
35-40	6-14	.38				
40-45	6-15	.28				
45-50	6-16	.30				
50-55	6-17	.50				
55-60	6-18	.55	6.7	8.41		

* less than

Assays of Drill Hole No. 16 — Depth 35'

Interval	Sample No.	% Ni	% Fe	% Mg	% Cr2O3	% Co
0-5	4-11	0.92	29.8	6.26	2.46	0.27
5-10	4-12	.86	14.6			
10-15	4-13	.33				
15-20	4-14	.41				
20-25	4-15	.44				
25-30	4-16	.29				
30-35	4-17	.32				

Assays of Drill Hole No. 17 — Depth 30'

Interval	Sample No	%Ni	% Fe	%Mg	%CrO$_3$
0-5	7-10	0.30	16.6		
5-10	7-11	1.36	16.8	7.32	0.02*
10-15	7-12	1.16	16.4		.07
15-20	7-13	.61	10.8		
20-25	7-14	.57			
25-30	7-15	.56	7.0		

* less than

Assays of Drill Hole No. 21 — Depth 20'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$
0-5	8-20	0.62			
5-10	8-21	.48			
10-15	8-22	.41			
15-20	8-23	.51			

Assays of Drill Hole No. 18 — Depth 25'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$
0-5	7-17	0.48			
5-10	7-18	.52			
10-15	7-19	.45			
15-20	7-20	.41			
20-25	7-21	.42			

Assays of Drill Hole No. 22 — Depth 35'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$
0-5	8-12	0.23			
5-10	8-13	.23			
10-15	8-14	.26			
15-20	8-15	.29	6.9		
20-25	8-16	.23			
25-30	8-17	.23	6.05		
30-35	8-18	.22			

Assays of Drill Hole No. 19 — Depth 20'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$
0-5	7-23	0.26			
5-10	7-24	.46			
10-15	7-25	.51			
15-20	8-1	.32			

Assays of Drill Hole No. 23 — Depth 30'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$	%Co
0-5	9-19	0.87	38.0	5.7	1.82	0.29
5-10	9-20	2.43	14.4			
10-15	9-21	1.07	10.6			
15-20	9-22	1.29	9.8	17.2	0.77	.17
20-25	9-23	1.16	10.0			
25-30	9-24	.61	6.6			

Composite sample of hole 23 is 0.01 % Mg

Assays of Drill Hole No. 20 — Depth 40'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO$_3$	%Co
0-5	8-3	0.98	41.4			
5-10	8-4	.83	41.6	6.47		
10-15	8-5	.98	36.0		1.06	0.11
15-20	8-6	1.22	20.2		.80	.10
20-25	8-7	.73	13.2			
25-30	8-8	.39				
30-35	8-9	.42				
35-40	8-10	.29	7.6			

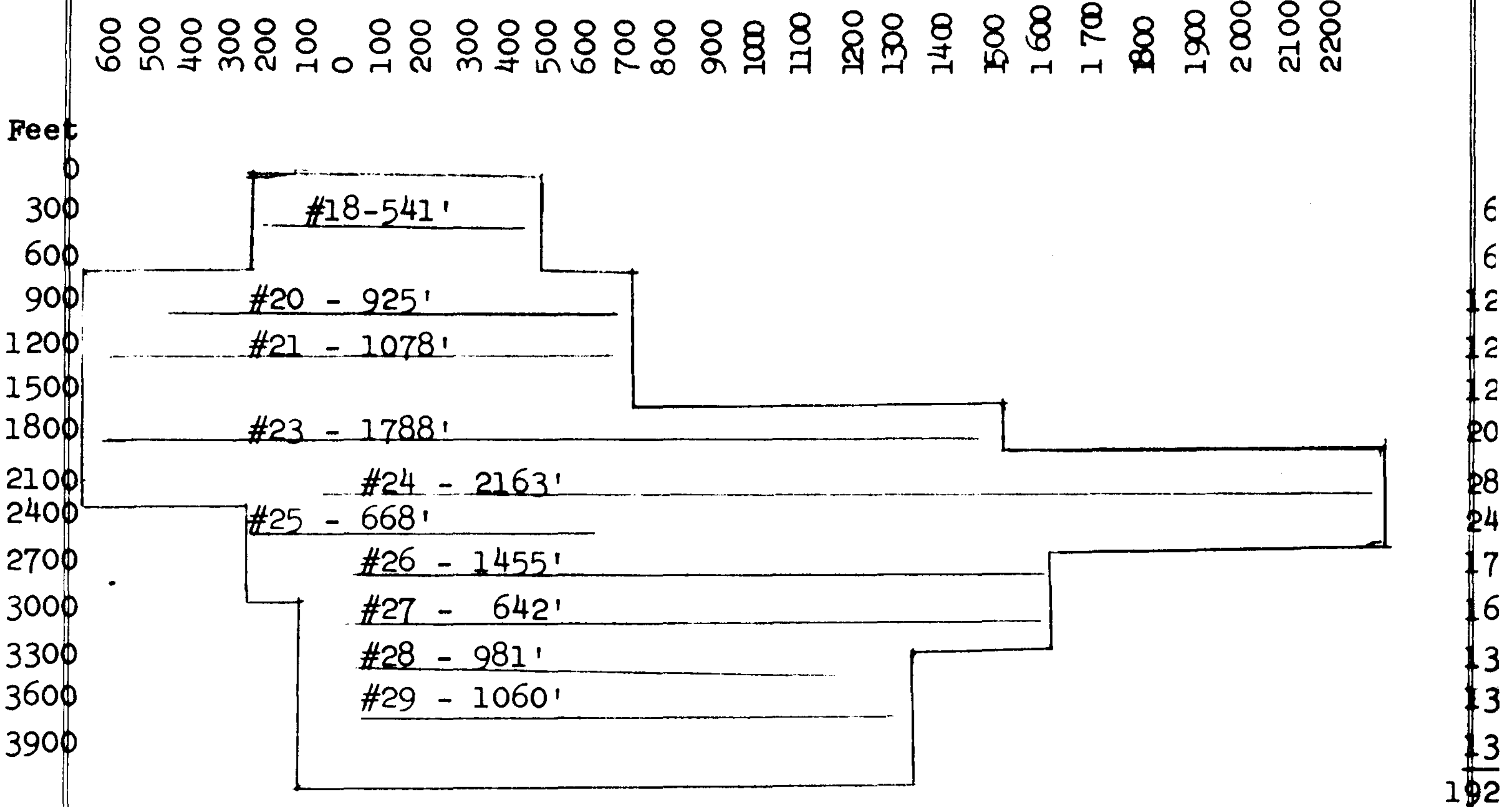
1.00% Ni

4. Work by Pacific Nickel

Method of Sampling

Bulldozer trenches were excavated to depths of 16 feet. Samples were taken at 10 foot intervals along the trenches in vertical cuts. No samples were taken where the trench was less than 5 feet deep. Samples varied in weight from about 5 to 10 pounds each, dependent upon depth of trench.

AREA COVERED ON RED FLATS FROM TRENCH #18 TO #29



Each square block = 100 x 300 feet = 30,000 sq. ft.

192 blocks x 30,000 sq. ft. = 5,''60,000 sq. ft. of area

5,760,000 x 8.745 = 50,371,200 c bic feet

at 16.2 cubic feet per ton = 3,109,395 tons

Average depth of trenches, types of ores and assays:

	Top Soil	Laterite & Float	Unconsolidated & Peridotite	Total Depth	Assay averages K.D. & E. % Ni	K.D. & Ledoux % Ni
#18	1.50	1.50	5.80	8.80	0.64	0.80
#20	1.45	5.15	2.80	9.40	0.97)	
#21	1.40	5.30	2.90	9.60	0.98)	
#23	1.23	3.78	2.87	7.88	0.61)	0.87 552 0.65 582 0.64 612 0.77 642 0.97
#24	0.63	3.04	5.44	9.11	0.86)	
#25	1.20	3.20	3.00	7.40	0.82)	
#26	0.80	2.70	5.30	8.80	1.04)	
#27	0.72	3.02	5.07	8.81	.81)	
#28	0.58	3.49	3.74	7.81	1.08	1.12
#29	0.66	3.63	5.65	9.94	0.95	0.95
Aver.	1.01	3.48	4.26	8.75	0.88	0.94

AVERAGE OF INDIVIDUAL AND COMPOSITE ASSAYS BY SEPARATE TRENCHES

Averages of Assays

Trench	Samples	(K.D. & E.)	Ledoux & K. D.
E & W-18	(12 samples)	0.64% Ni	0.80 (Composite #1-10 samples)
E & W-20	(20 samples)	0.97%	
E & W-21	(28 samples)	0.98%	
*E & W-22	(3 samples)	0.85%	(Composite #4-43 samples)
E & W-23	(41 samples)	0.61%	0.87 (Composite #5-23 samples)
E-24	(40 samples)	0.86%	(Composite #6-21 samples)
E & W-25	(16 samples)	0.82%	
E-26	(17 samples)	1.04%	
E-27	(13 samples)	0.81%	
E-28	(19 samples)	1.08%	1.12 (Composite 'A' & B - 17 samples)
E-29	(24 samples)	0.95%	0.95 (Composite A,B & C - 24 samples)
Average		0.87%	0.94
Average by number of samples		0.86%	0.91
Length of trench x assay		0.99%	slide rule calculations
Length x depth x assay		0.87%	slide rule calculations

K.D. & E. - 233 samples - 160 assays
Ledoux & K.D. - 138 samples - 9 assays

*Omitted from general average due to shallow trenching

CALCULATIONS FOR COMPUTING TONNAGE USING SPECIFIC GRAVITY DETERMINATIONS

Specific gravity tests

Laterite

59.00 grams displaced 18.1 ml water = 3.26 sp. gr.

Serpentine

51.25 grams displaced 19.9 ml water = 2.58 sp. gr.

Tonnage calculations:

	Depth in ft.	%	Sp. gr.
Top soil & laterite	4.488	51.3	3.26
Serpentinized peridotite	4.257	48.7	2.58
	8.745	100.0	2.93

	% Wt.	Cubic ft./ton
Estimated free water	25.0%	32.0
Dry ore (Sp.gr. 2.93)	75.0%	10.9
	100.0%	16.2

50,371,200 cubic feet in area sampled ÷ 16.2 cubic feet per ton
= 3,109,395 tons.

AVERAGE OF THE ASSAYS FROM EACH TRENCH

						Trench Average
E-18-45-347	8 assays	-	Average	0.80% Ni	(6-E-2-KD)	
W-18-44-134	4 assays	-	Average	0.38%	(E)	0.64%
E-20-66-530	11 assays	-	Average	0.79%	(E)	
W-20-85-315	9 assays	-	Average	1.19%	(K-D)	0.97%
E-21-65-555	14 assays	-	Average	0.79%	(12-E-2-KD)	
W-21-83-513	14 assays	-	Average	1.16%	(K.D.)	0.98%
E-22-52-82	2 assays	-	Average	0.92%	(K.D.)	
W-22-100	1 assay	-	Average	0.72%	(E)	0.85%
E-23-86-656	18 assays	-	Average	0.69%	(E)	
E-23-1054-1324	1 assay	(10 sample composite)		0.65%	(K.D.)	
W-23-64-454	13 assays	-	Average	0.46%	(E)	0.61%
E-24-63-2143	24 assays	-	Average	0.86%	(15-E-9-KD)	
E-24-653-883	1 assay	(3 sample composite)		0.95%	(KD)	
E-24-1123-1333	1 assay	(8 sample composite)		0.88%	(KD)	
E-24-1873-2143	1 assay	(5 sample composite)		0.78%	(KD)	0.86%
E-25-97-467	11 assays	-	Average	0.76%	(6-KD-5-E)	KD 0.90%
W-25-51-171	5 assays	-	Average	0.94%	(KD)	E & KD 0.82%
E-26-380-560	7 assays	-	Average	0.44%	(E)	
E-26-955-1165	1 assay	(8 sample composite)		1.24%	(KD)	(Average
E-26-1195-1435	1 assay	(9 sample composite)		1.33%	(KD)	E & KD 1.04%)
E-27-82-642	13 assays	-	Average	0.81%	(9-KD-4-E)	0.81%
E-28-80-441	1 assay	(10 sample composite)		1.28%	(KD)	
E-28-611-901	1 assay	(7 sample composite)		0.88%	(KD)	
E-28-931-961	2 assays	-	Average	0.77%	(1-KD-1-E)	1.08%
E-29-110-320	1 assay	(8 sample composite)		0.83%	(KD)	
E-29-440-710	1 assay	(9 sample composite)		0.94%	(KD)	0.95%
E-29-890-1060	1 assay	(7 sample composite)		1.09%	(KD)	

AVERAGE OF INDIVIDUAL ASSAYS BY KENNARD & DRAKE

E-18-317 - 0.97%
E-18-347 - 0.88%

W-20- 85 - 1.22%
W-20-135 - 1.02%
W-20-165 - 1.07%
W-20-195 - 1.38%
W-20-225 - 1.25%
W-20-255 - 1.39%
W-20-285 - 1.55%

W-21- 83 - 1.14%
W-21-113 - 1.21%
W-21-143 - 1.01%
W-21-173 - 1.04%
W-21-203 - 1.12%
W-21-233 - 1.23%
W-21-263A - 1.00%
W-21-293 - 1.05%
W-21-313 - 1.50%
W-21-393 - 1.53%
W-21-513 - 1.23%

E-21-525 - 1.04%
E-21-555 - 0.87%

E-22- 52 - 1.01%
E-22- 82 - 0.84%

E-23-357 - 0.82%
E-23-387 - 0.84%
E-23-447 - 1.12%
E-23-477 - 1.06%

E-24-213 - 1.20%
E-24-243 - 1.47%
E-24-373 - 1.28%
E-24-403 - 0.46%
E-24-433 - 1.22%
E-24-503 - 0.64%
E-24-593 - 1.24%
E-24-943 - 0.61%
E-24-1933 - 1.01%
E-24-2053 - 1.01%

W-25- 51 - 1.38%
W-25- 81 - 0.91%
W-25-111 - 0.83%
W-25-141 - 0.81%
W-25-171 - 0.79%

E-25- 97 - 0.93%
E-25-127 - 0.99%
E-25-157 - 0.83%
E-25-177 - 0.77%
E-25-287 - 0.84%
E-25-317 - 0.98%

E-27- 82 - 1.10%
E-27-152 - 1.12%
E-27-242 - 0.75%
E-27-332 - 0.73%
E-27-382 - 0.91%
E-27-412 - 0.87%
E-27-552 - 0.65%
E-27-612 - 0.77%
E-27-642 - 0.97%

E-28-961 - 0.91%

59.94%

59 assays

Average 1.02%

552 0.65
582 0.84
612 0.77
642 0.97

AVERAGE OF INDIVIDUAL ASSAYS BY EISENHAUER

W-18- 44	-	0.35%
W-18- 74	-	0.44%
W-18-104	-	0.42%
W-18-134	-	0.32%
E-18- 45	-	1.11%
E-18- 75	-	0.74%
E-18-105	-	0.76%
E-18-135	-	1.01%
E-18-257	-	0.22%
E-18-287	-	0.70%
W-20-315	-	0.94%
E-20- 66	-	0.86%
E-20-260	-	0.78%
E-20-290	-	0.70%
E-20-320	-	0.99%
E-20-350	-	0.93%
E-20-380	-	0.80%
E-20-410	-	0.73%
E-20-440	-	0.88%
E-20-470	-	0.75%
E-20-500	-	0.56%
E-20-530	-	0.70%
W-21-423	-	1.25%
W-21-453	-	1.04%
W-21-483	-	0.95%
E-21- 65	-	0.79%
E-21- 95	-	0.93%
E-21-125	-	0.94%
E-21-155	-	1.17%
E-21-175	-	0.95%
E-21-315	-	0.50%
E-21-345	-	0.53%
E-21-375	-	0.66%
E-21-405	-	0.90%
E-21-435	-	0.65%
E-21-465	-	0.62%
E-21-495	-	0.49%
W-22-100	-	0.72%

W-23- 64	-	0.88%
W-23- 94	-	0.43%
W-23-124	-	0.56%
W-23-154	-	0.55%
W-23-214	-	0.64%
W-23-244	-	0.22%
W-23-274	-	0.27%
W-23-304	-	0.46%
W-23-334	-	0.32%
W-23-364	-	0.46%
W-23-394	-	0.30%
W-23-424	-	0.54%
W-23-454	-	0.31%
E-23- 86	-	0.55%
E-23-116	-	0.35%
E-23-146	-	0.92%
E-23-166	-	0.26%
E-23-266	-	0.47%
E-23-296	-	0.53%
E-23-326	-	0.47%
E-23-506	-	0.63%
E-23-536	-	0.94%
E-23-566	-	0.73%
E-23-596	-	0.71%
E-23-626	-	0.64%
E-23-656	-	0.76%
E-24- 63	-	0.74%
E-24- 93	-	0.94%
E-24-123	-	0.77%
E-24-273	-	0.87%
E-24-463	-	0.84%
E-24-443A	-	0.98%
E-24-473A	-	0.72%
E-24-533	-	0.69%
E-24-563	-	0.59%
E-24-623	-	1.07%
E-24-853	-	0.65%
E-24-913	-	0.57%
E-24-1903	-	0.[illegible]
E-24-1963	-	0.77%
E-24-2083	-	0.93%

E-25-347	-	0.44%
E-25-377	-	0.86%
E-25-407	-	0.46%
E-25-417	-	0.70%
E-25-437	-	0.64%
E-25-467	-	0.62%
E-26-380	-	0.35%
E-26-410	-	0.40%
E-26-440	-	0.40%
E-26-470	-	0.41%
E-26-500	-	0.47%
E-26-530	-	0.56%
E-26-560	-	0.48%
E-27-212	-	0.72%
E-27-272	-	0.73%
E-27-442	-	0.62%
E-27-582	-	0.64%
E-28-931	-	0.55%
		64.44

97 samples

Average 0.66%

AVERAGE DEPTH OF THE THREE TYPES OF ORES FROM INDIVIDUAL TRENCHES

Note: All parts of trenches less than five feet deep omitted in the following calculations.

	TOP SOIL	LATERITE & FLOAT	UNCONSOLIDATED & PERIDOTITE	TOTAL DEPTH
W-18- 24-144	1.7	0.3	8.0	10.0
E-18- 25-175	1.5	2.7	3.2	7.4
E-18-227-397	1.2	1.4	6.2	8.8
	4.4	4.4	17.4	26.2
Average	1.5	1.5	5.8	8.8
W-20- 45-355	1.6	3.5	3.7	8.8
E-20-240-570	1.3	6.8	1.9	10.0
	2.9	10.3	5.6	18.8
Average	1.45	5.15	2.80	9.40
E-21- 45-195	1.3	4.8	4.1	10.2
E-21-255-565	1.5	7.9	0.3	9.7
W-21- 73-513	1.3	3.2	4.2	8.7
	4.1	15.9	8.6	28.6
Average	1.4	5.3	2.9	9.6
W-23- 54-454	1.5	3.5	3.5	8.5
E-23- 86-176	1.5	4.5	1.5	7.5
E-23-256-336	1.5	3.9	0.4	5.8
E-23-347-487	1.5	2.4	2.3	6.2
E-23-496-676	1.1	5.0	3.1	9.2
E-23-1034-1334	0.3	3.4	6.4	10.1
	7.4	22.7	17.2	47.3
Average	1.23	3.78	2.87	7.88
E-24- 43-133	0.7	6.3	1.5	8.5
E-24-203-293	0.4	0.7	8.2	9.3
E-24-353-483	1.1	1.3	8.4	10.8
E-24-443A-653	0.9	2.7	5.4	9.0
E-24-803-953	0.6	4.9	2.7	8.2
E-24-1153-1353	0.2	2.0	7.9	10.1
E-24-1853-2163	0.5	3.4	4.0	7.9
	4.4	21.3	38.1	63.8
Average	0.63	3.04	5.44	9.11

	TOP SOIL	LATERITE & FLOAT	UNCONSOLIDATED & PERIDOTITE	TOTAL DEPTH
E-26-370-590	1.3	3.2	3.0	7.5
E-26-905-1455	0.3	2.2	7.6	10.1
	1.6	5.4	10.6	17.6
Average	0.8	2.7	5.3	8.8
E-27- 62-262	0.500	2.237	5.000	7.737
E-27-322-442	0.808	3.077	4.808	8.693
E-27-552-642	0.850	3.750	5.400	10.000
	2.158	9.064	15.208	26.430
Average	0.719	3.021	5.069	8.810
E-28- 70-210	0.357	3.071	4.964	8.392
E-28-301-451	0.467	1.038	7.731	9.236
E-28-601-731	0.604	6.437	None	7.041
E-28-861-981	0.900	3.400	2.250	6.550
	2.328	13.946	14.945	31.219
Average	0.582	3.487	3.736	7.805
E-29-100-320	0.159	3.545	6.432	10.136
E-29-430-560	0.375	7.458	1.500	9.333
E-29-610-710	1.200	1.300	6.400	8.900
E29-880-1060	0.895	2.211	8.263	11.369
	2.629	14.514	22.595	39.738
Average	0.657	3.628	5.649	9.934
E-16-06-256	0.66	0.44	8.60*	9.70
W- 9-110-270	0.86	0	9.40*	10.26
W- 9-250-490	0.60	1.50	8.20*	10.30
W- 9-630-830	0.85	0.14	9.81*	10.80
	2.31	1.64	27.41	31.36
Average	0.77	0.55	9.14	10.46
W-25- 51-181	0.9	1.5	3.9*	6.3
E-25- 77-197	1.4	2.4	3.8*	7.6
E-25-287-487	1.4	5.8	1.2*	8.4
	3.7	9.7	8.9	22.3
Average	1.2	3.2	3.0	7.4

* Unconsolidated serpentinized peridotite and peridotite.

AVERAGE DEPTH IN FEET

	Top Soil	Laterite & Float	Unconsolidated serpentinized peridotite and peridotite	Total Depth	% Ni
9-W-110-270	0.86	0	9.40	10.26	0.47
9-W-250-490	0.60	1.50	8.20	10.30	
9-W-630-830	0.85	0.14	9.81	10.80	0.37
	2.31	1.64	27.41	31.36	
Average	0.77	0.55	9.14	10.46	0.42
Average by number of samples					0.44

W-9 - Composite 'A' 0.47%Ni - 13 samples - (120-470)
W-9 - Composite 'B' 0.37%Ni - 7 samples - (630-810)

E-16-06-256	0.66	0.44	8.60	9.70	0.42

E-16 - Composite 'A' 0.42%Ni - 9 samples - (16-256)

Note: The above trenches #9 and #16 were not included in the tonnage estimates.

IV. LAND AND LEGAL

The land upon which the Pacific Nickel Corporation deposits are located has not been surveyed by the Government so as to have available a description by section, township and range, nor is there a metes and bounds survey available at this time. Involved are the following unpatented mining claims located in Curry County, Washington, which were acquired by assignment of contracts with the locators or the successors to the locators of the claims. The recording data hereinafter set forth is located in the official Mining Records of said county and state:

Claim No.	Date of Filing	Volume and Page No.	
1	July 1, 1936	10	510
2	July 1, 1936	10	511
3	July 1, 1936	10	512
4	July 1, 1936	10	512
5	July 1, 1936	10	513
6	July 1, 1936	10	514
7	July 1, 1936	10	514
8	July 1, 1936	10	516
9	July 1, 1936	10	526
Relocation Notice	July 13, 1945	12	321-322
10	April 14, 1941	11	356

(See Gold Beach Quadrangle Map in back pocket)

METALLURGY

A. Report of Preliminary Beneficiation Tests Conducted on Red Flats Nickeliferous Serpentine Ore by Southwestern Engineering Company, Los Angeles, California

SWECO LOT NO. 3763-Sp., 3810, 3815, 3867

Samples

The samples used for the tests reported herein consisted either of the rocky fraction of the ore or represented ore predominating in so-called serpentine as noted from visual inspection at the time the samples were cut. This so-called serpentine perhaps is more accurately described as serpentinized peridotite frequently occurring in unconsolidated and fractured condition. While it may be mixed with laterite, in general, it underlies the latter and rests upon the unaltered basic rock, peridotite. This fractured serpentinized peridotite usually is somewhat brown or yellow in color due to oxidized iron minerals contained in the fractures. This material is characterized by a high magnesia content as compared with laterite.

SWECO Lot No.	% Ni	
3763-Sp.	0.801*	Representing rock portion of a 1000 lb. grab sample.
3810	0.786*	The + 20 mesh fraction of serpentine samples from trenches 24 and 27 after crushing to -1/4".
3815	0.48	Composite of 39 low grade serpentine samples from trenches W-9, E-16 and E-23.
3867	0.91	Representing individual samples noted as consisting largely of serpentinized rock from trenches 18, 20, 21 and 23.

*Average calculated head

Object of Tests

The purpose of the tests reported herein was to try to beneficiate the so-called serpentine ore in respect to its nickel content. It was thought that beneficiation of the nickel might also be accompanied by some beneficiation of the iron content. It was also thought that a product beneficiated in respect to nickel and iron might have a lower tenor of magnesia which would be important from the standpoint of the Krupp-Renn process.

An important objective would be to try to beneficiate so-called serpentine ore of borderline or subgrade nickel content in order to bring it within the range treatable by the Krupp-Renn process.

Most of the tests reported herein were along lines of a reducing roast followed by magnetic separation. This seemed to be a logical approach as metallic nickel, metallic iron and certain oxides of the latter are known to be magnetic. Both metallic nickel and metallic iron form at relatively low temperature.

Very recently, interest has arisen in possibilities of effecting some beneficiation of the raw ore either through upgrading of the nickel-iron content or through eliminating a portion of the magnesia by desliming, flotation or other means.

RED FLATS NICKELIFEROUS SERPENTINE ORE

RESULTS OF PRELIMINARY REDUCING ROASTING AND MAGNETIC SEPARATION TESTS

SWECO		Calculated head		Magnetic conct.		Magnetic tailing		% Distribution		Ratio of conctr.	Roast		Ore		Coal		Re-		No. of
Lot No.	Test No.	% Wt.	% Ni	% Wt.	% Ni	% Wt.	% Ni	Conct. Ni	Tail. Ni	by wt.	Temp. F	Hours	gms.	mesh	gms.	mesh	grind mesh	Mag. Sep.	cleanings
3763-Sp.	1	98.50	0.787	36.50	1.17	62.00	0.58	54.28	45.72	2.74:1	2050	2	100.0	-¼"	30	-16	-65	Wet	2
3763-Sp.	2	102.00	0.857	34.00	1.20	68.00	0.66	47.62	52.38	2.94:1	1500	1½	100.0	-¼"	30	U.P. -¼"	-65	Wet	0
3763-Sp.	3	88.50	0.851	47.00	1.22	41.50	0.67	67.34	32.66	2.13:1	1500	1½	100.0	-35	30	-16	-35	Dry	0
3763-Sp.	4	97.51	0.886	50.63	1.21	46.88	0.58	69.11	30.89	1.98:1	1700	1½	200.0	-¼"	30	-16	-16	Dry	2
3763-Sp.	4-A	100.00	0.571	45.52	0.86	54.48	0.33	68.53	31.47	2.20:1								Dry	2
3763-Sp.	4&4-A	97.51	0.834	62.01	1.15	35.50	0.35	85.17	14.83	1.60:1								Dry	2
3763-Sp.	5	86.44	0.682	42.06	1.04	44.38	0.55	64.19	35.81	2.38:1	1700	1½	400.0	-¼"	60	-16	-35	Wet	1
3763-Sp.	6	91.50	0.793	45.25	1.15	46.25	0.59	65.60	34.40	2.16:1	1700	1	400.0	-¼"	60	-16	10 M.	Wet	1
3810	1	92.00	0.775	22.75	1.25	69.00	0.71	36.73	63.27	4.40:1	2000	2	100.0	-8	20	-16	-65	Wet	2
3810	2	96.50	0.847	58.00	1.10	38.00	0.55	75.32	24.68	1.72:1	2300	2	100.0	-8	30	-16	-80	Wet	2
3810	3	101.50	0.757	28.30	1.19	83.50	0.65	28.30	71.70	3.54:1	1500	1	100.0	-8	30	-16	-65	Wet	1
3810	4	91.75	0.765	31.50	1.09	60.25	0.70	44.88	55.12	3.17:1	2100	1	100.0	-100	30	-16	-100	Wet	2
3867	1	99.00	0.857	64.00	1.12	35.00	0.40	83.66	16.34	1.56:1	1700	1½	100.0	-¼"	30	-16	-16 & -35	Dry	2
3867	2	96.50	0.849	55.50	1.16	41.00	0.50	75.85	24.15	1.80:1	1800	1½	100.0	-¼"	30	-16	-16 & -35	Dry	2
3867	3	94.50	0.872	57.50	1.15	37.00	0.57	75.82	24.18	1.74:1	1900	1½	100.0	-¼"	30	-16	-16 & -35	Dry	2

Summary of Preliminary Beneficiation Tests

The preliminary tests conducted involving a reducing roast and either wet or dry magnetic separation resulted in beneficiation in all cases in respect to the nickel content. Microscopic inspection of reduced calcine after introduction of a nickel indicating solution indicates that the nickel in the so-called serpentine is not distributed uniformly throughout the rock but is somewhat concentrated in certain area, streaks and spots while others contain less or little nickel.

It is believed that optimum conditions for carrying out the reducing roasts and magnetic separation probably were not attained in these tests. No conclusions have been reached as to the advantages of wet as compared to dry magnetic separation in this case.

Results of various tests are reported in the attached tabulation and average (arithmetical) results of three groups of tests are shown below:

Sweco Lot No.	Number of Tests	Calculated Head % Ni	Magnetic Concentrate % Wt.	Magnetic Concentrate % Ni	Tailing % Wt.	Tailing % Ni	% Distribution Mag. Conct. Ni	% Distribution Tailing Ni
3763-Sp.	6	0.801	44.47	1.16	49.60	0.57	64.03	35.97
3810	4	0.786	35.14	1.16	62.69	0.65	46.31	53.69
3867	3	0.859	59.00	1.14	37.67	0.49	78.44	21.56

Results of individual tests representing best nickel recovery are given below:

Sweco Lot No.	Test No.	Calculated Head % Ni	Magnetic Concentrate % Wt.	Magnetic Concentrate % Ni.	Tailing % Wt.	Tailing % Ni	% Distribution Mag. Conct. Ni	% Distribution Tailing Ni
3763-Sp.	4 & 4-A	0.834	62.01	1.15	35.50	0.35	85.17	14.83
3810	2	0.847	58.00	1.10	38.00	0.55	75.32	24.68
3867	1	0.857	64.00	1.12	35.00	0.40	83.66	16.34

The foregoing results are considered to be sufficiently encouraging to justify much more comprehensive investigation.

In general, there appears to be a relationship between the weight of magnetic concentrate and nickel recovery. The lower average nickel recovery for the tests on Lot 3810 seems largely due to efforts to produce low weight of magnetic concentrate in tests #1, #3 and #4.

Highly preliminary experiments thus far conducted in investigating the possibilities . beneficiating the raw (unroasted) so-called serpentine ore indicate that scrubbing and desliming will yield a slime product somewhat upgraded in nickel (probably also in iron, silica and alumina) and somewhat downgraded in magnesia. Means for upgrading the sand fraction will need to be developed and the possibilities of gravity concentration, flotation and direct magnetic separation as well as reducing roasting and magnetic separation should be investigated on this portion of the ore.

Highly preliminary direct flotation experiments on total low grade crude serpentine ore using a fatty acid reagent resulted in slight upgrading of nickel in the concentrate and slight downgrading in magnesia. However, this work has not as yet progressed sufficiently to be very indicative. The hope would be to eliminate magnesia by either flotation or depression. The possible success of this approach would seem to depend upon whether a fraction of the serpentine, which is magnesium silicate, has a different content of magnesia or silica due to its altered condition.

Reducing Roasting Procedure

In all of the preliminary reducing roasts reported herein, the ore was mixed with bituminous coal, then placed in either a graphite or a clay crucible and the roast was conducted in an electrically heated muffle furnace. In a number of trials the crucible was withdrawn at the end of the roast and covered with a loose fitting cover and allowed to cool. In other trials the partially cooled charge was quenched in water. In last trials a small stream of natural gas was introduced into the crucible while the charge was cooling. All of these procedures gave an opportunity for reoxidation, that employing gas probably gave the least.

It is considered that the procedures employed in these preliminary laboratory experiments would not necessarily be followed in practical application. The reducing roasting of nickeliferous ores has been quite well developed on a large scale in Cuba and elsewhere and possibly such procedures or modifications thereof might be applicable to this problem. It is believed that a reducing roast for rough concentration by magnetic separation could be much less critical and not necessarily as thorough as one required for subsequent ammonia leaching.

In Cuba, the ores are ground to about 60 mesh before roasting. In the preliminary tests reported herein roasting at -1/4" seemed to be as effective as at a finer size.

B. Krupp-Renn Process

1. History

In the late 1920's a small pilot plant was erected in Magdeburg, Germany for the treatment of low grade iron ores and nickeliferous iron ores by the Krupp-Renn Process.

In the year 1934, Krupp built a Renn plant near Frankenstein in Silesia with a kiln of 11.8 feet x 164 feet long, for the conversion of nickel bearing ores (Garnierites) into nickel bearing iron pellets. This plant was in operation without interruption from 1935 to 1945. It was enlarged in 1942 by the erection of a second kiln of the same size. On the average each kiln was operating 300 days per year and treating 80,000 tons of ore per year. This plant was dismantled and moved to the U.S.S.R. in 1947. During the operation of this plant, Ing. Harald Timmermann of Krupp directed the project and learned much concerning the treatment of nickel oxide ores.

By 1956 the first kiln for converting nickeliferous ores of Larymna, Greece into an iron nickel alloy was built by the Societe Anonyme Hellenique de Produits & Engrais Chimiques. The ore ran as follows:

Ni	1.4 - 1.8%
Fe	25 - 36%
S_1O_2	15 - 33%
AL_2O_3	11 - 15%
CaO	2 - 3%
MaO	2 - 3%

Utilizing coke breeze and anthracite as fuel the kiln, which is 13 feet in diameter and 300 feet long, is treating 400 tons of ore a day and producing approximately 425 tons of luppen which runs:

Fe	90%
Ni	4%
Cr	1.9%
Sulfur	.3%

In addition, another unit is now being prepared to be placed in operation in the very near future. The lining in the hot zone of the first kiln lasted for seven months and fourteen days. It is expected that this lining life may be materially increased.

In the history of the Krupp organization, sufficient know-how from the operation of the Frankenstein and Larymna plants is available to warrant the utilization of this process in areas where it is applicable throughout the world.

2. Application of the Process

The application of the Krupp-Renn Process to the ores of the Red Flat District near Gold Beach, Oregon depends on further study.

The utilization of the process is limited to areas where high magnesia material is not encountered. If the slag in the Renn kiln exceeds 27% MgO it becomes very difficult to work the process according to Ing. Timmermann. The slag at this point of high magnesia content becomes very viscous. It is therefore important that the magnesia content of the feed be kept at a reasonable figure.

In approaching the problem of the treatment of the Red Flats ore, a low

MgO content in the slag has been attained by the mixing of one part of lateritic material with one part serpentine. Such a mixture, based on the information we have at this time, would appear to be satisfactory as feed to the Renn furnace. However, this supposition must be proved by further pilot plant tests both in the United States and in Germany.

It also appears that the ore must be dried prior to introduction into the Renn kiln in order to increase kiln capacity. It is felt on the basis of work done in Cuba that such drying could be carried out for approximately $1.00 per ton.

The nickeliferous laterites and underlying serpentines of the Red Flat area would produce material of approximately the following analysis. This information is based on Pacific Nickel work and the U. S. Bureau of Mines "Report of Investigation 5072 - Preliminary Investigation of the Red Flats Nickel Prospect".

a.	Laterite			Serpentine	Dry Basis
	Fe	=	29.0 %	10.0%	Water (Free) 15%
	Ni	=	.91%	.91%	
	Co	=	.10%	.10%	
	Al_2O_3	=	6.0 %	2.0 %	
	Cr_2O_3	=	2.5 %	1.0 %	
	CaO	=	1.1 %	1.2 %	
	MgO	=	5.5 %	25.0 %	
	SiO_2	=	26.0 %	40.0 %	
	S	=	.04%	.04%	
	P	=	.03%	.03%	
	O_2	=	12.0 %	4.0 %	
	Loss on Ignition	=	11.4 %	10.0 %	

Due to high magnesia content of serpentine it is necessary to blend laterite with serpentine to produce an acceptable feed. We shall assume a ratio of one to one for this study.

Constituent	Fe	Ni	Co	Al_2O_3	Cr_2O_3	CaO	MgO	SiO_2	S	P	O_2	L.O.I.
Serpentine	10.0	.91	.10	2.0	1.0	1.2	25.0	40.0	.04	.03	4.0	10.0
Laterite	29.0	.91	.10	6.0	2.5	1.1	5.5	26.0	.04	.03	12.0	11.4
Combined	39.0	1.82	.20	8.0	3.5	3.3	30.5	66.0	.08	.06	16.0	21.4
Mill Feed Average	19.5	.91	.10	4.0	1.8	1.1	15.2	33.0	.04	.03	8.0	10.7

Plant Feed		Flux	Slag	Slag %
Fe	19.5 %			
Ni	.91%			
Co	.10%			
Al_2O_3	4.0 %	5%	9.0	15.0 %
Cr_2O_3	1.8 %		1.4	2.34
CaO	1.1 %		1.1	1.84
MgO	15.2 %		15.2	25.46
SiO_2	33.0 %		33.0	55.36
S	.03%		59.7	100.00%
P	.04%			
O_2	8.0 %			
L.O.I.	10.7 %			

3. Fuel

Reduction fuel for the Renn process must be material with volatile combustible matter of 10% or less. Since anthracite and coke breeze is unavailable at a reasonable price, on the West Coast, it is necessary to subject available fuels to low temperature carbonization.

The nearest available fuels are located two townships north of Gold Beach between Marshfield, Oregon and Coquille, Oregon. This coal would be of sub-bituminous or lignite rank. Friability of the coals appears to range around 27%. The slag index is around 25%. Agglomeration tests on samples indicate that the coals are non-coking in character and produce a non-coherent granular char rather than coke when subjected to low temperature carbonization.

A low temperature carbonization test was made on a large sample of coal from the Southport Mine near Marshfield, Oregon and was submitted by the State Department of Geology & Mineral Industries from Oregon to the American Lurgi Corporation in 1944. Each ton of coal produced a char of 62.4%, tar oil of 7%, and gas of 12%, with water 18.6%. It is expected that this yield could be obtained from other coals in the Coos Bay District and that sufficient tonnages are probably available in this district to be utilized in the operation of a Renn plant near Gold Beach, Curry County, Oregon.

An extremely thorough study has been made of the Geology & coal resources of the Coos Bay Quadrangle in Oregon by John Eliott Allen and Ewart M. Baldwin. This study is known as Bulletin No. 27 and has been issued by the State of Oregon, Department of Geology & Mineral Industries, 702 Woodlark Building, Portland 5, Oregon.

We have used the information in that bulletin to arrive at figures for fuel costs.

4. Product

If our assumptions concerning the materials which will be fed to this process are borne out by further investigation, the following product can be expected.

The recovery of nickel in this process is generally in excess of 90% of the contained nickel and 92% of the contained iron.

Assume ore Fe equals 19.5% : Recovery 92% : Product Fe 91% :

$$\frac{91\%\ \text{Fe in product} \times 100}{92\%\ \text{recovery} \qquad 19.5\%\ \text{Fe in ore}} \quad \text{equals } 5.07 \text{ tons ore/ton luppen}$$

Thus the final product will contain:

$$\frac{90\%}{100} \times .91\ \text{Ni} \times 5.07 \text{ equals } 4.15\%\ \text{Ni}$$

or 84# of nickel per ton of luppen

The luppen will assay approximately as follows, if the foregoing raw materials are used:

Fe	91.0%
Ni	4.15%
C	.8 %
Cr_2O_3	.1 %
SiO_2	2.0 %
MgO	1.0 %
Al_2O_3	.4 %
S	Not available from present data
P	Not available from present data

VI MARKET

A market analysis concerning the sale of nickeliferous luppen has not been conducted in the United States.

The following data is available concerning the sale of this material from Friedrich Krupp in Essen, Germany.

The Greek nickeliferous ore treatment plant at Larymna was able to dispose of its product as previously shown to a French concern utilizing it in the manufacture of stainless steel. The last contract noted (February, 1957) was for 5,000 metric tons of product at a value of $2.00 per pound of contained nickel.

International Nickel Company offers nickel F.O.B. refinery at Port Calbourne, Ontario at 74 cents per pound of contained nickel. There are several contracts by the U. S. Government for the purchase of nickel in form of ferro-nickel. These prices range up to $1.42 per pound of contained nickel.

The introduction of this material to the U. S. market would bring about certain changes in steel making procedure. However it is believed that at this time there are no insuperable obstacles to overcome in this particular respect.

If the nickeliferous luppen would be utilized as base stock for stainless steels, it would appear that the value of contained nickel in a nickel short market would be at least 74 cents a pound. The iron would probably be valued at 3 cents a pound.

VII ECONOMICS

A. Assumptions

1. In order to study the possible economics of this project certain basic assumptions must be made.

a. Ore deposit:

It is assumed that the ore deposits will have the values as estimated earlier in the section on ore reserves. These estimates can only be confirmed by further work on the ore deposit.

b. Process:

It is assumed that Krupp Renn process is applicable to the treatment of the Red Flat nickel ores. This assumption is based upon preliminary information and must be substantiated by means of pilot plant testing as well as additional data concerning the ore body.

c. Market:

It is assumed that a market for the product as outlined in the market section will be available in the world. The assumptions in this work are based upon conditions prevailing in March, 1957.

It is recommended that further work in the field of market analysis be done concerning disposition of this material.

d. Fuel:

It is assumed that a suitable fuel is available in the nearby area or available by means of ocean transport from Gold Beach, Oregon for this project. However at this moment adequate information concerning the fuel is not available and additional studies to determine that fuel supply must be undertaken in greater detail.

2. Cost of Product

With the foregoing assumptions in mind the economics of the application of the process to this particular ore body can be approximately determined.

B. Low Temperature Carbonization Computations

1. Coat With 40% Volatiles

Basic Variable Data	Assumptions
a. Cost per ton of coal at site	$5.40/ton
b. Cost of electric power - cents per kwh	$ .01/kwh
c. Cost of water - cents per 1000 gallons	$ .25/1000 gals
d. Cost of steam - price per ton	$1.82/ton
e. Cost of labor - labor - hourly rate	$2.30
foreman - hourly rate	$3.00
f. Market price of tar oil - produced - cents/gallon	$ .06/ga..

Basic Fixed Data

880 tons coal per day feed

516 tons coke per day produced

Operating 350 days per year

Connected load - 400 kw per hour

Water Consumption - 12,000 gallons per hour

Steam Consumption - 1.1 tons per hour

Heat Value of surplus gas - 3168 x 10^6 Btu per day

Repaires & Maintenance - 4% investment/year

Fringe Benefits on hourly wages - 12%

Supervision Salaries - 20% of labor

Amortization - 5% per year on investment

Interest - 5% per year on unpaid balance

2. 2 - 440 Ton Per Day Units

Basis: 880 tons coal

516 tons coke

350 days per year

	per day	per ton coke
I RAW MATERIAL		
880 tons coal		
5.40/t x 880 tons/day	4,752.00	9.22
Raw Material Cost	4,752.00	9.22
II OPERATING COSTS		
A. Power 400 kw connected load	96.00	.19
400 kw x 24 hrs. x .01		
B. Water 12,000 gals/hr. x		
24 hrs. x .25/1000 gals.	72.00	.14
C. Steam 1.1 tons/hr.		
1.1 t/hr. x 24 hrs. x 1.82/ton	48.00	.09
D. Repairs & Maintenance		
4% Investment/year		
$\frac{4\%}{100}$ x \$1,964,500 ÷ 350 days	222.00	.44
E. Wages - Labor 2.30/hr. -		
Supervision 3.00/hr.		
12% fringe benefits		
2.30/hr. x 8 hrs. x 16 men		
x 1.12 fr. ben. x $\frac{365}{350}$ days	353.00	.69

II OPERATING COSTS (Continued)	per day	per ton coke
E. (Cont'd)		
Foremen		
3.00/hr. x 8 hrs. x 3 men		
x 1.12 fr. ben. x $\frac{365}{350}$	86.20	.17
F. Salaries		
20% of wages		
$\frac{20\%}{100}$ x (86.20 + 353.00)	87.84	.17
Operating Cost Total	965.04	1.89
III CAPITAL COSTS		
A. Investment		
1. Plant Delivered		
German Port		
5,010,500 DM ÷ 4.20 DM/$	1,195,000	
2. Additional Construction		
$2,598.750 + 4.20 DM/$	619,500	
3. Coke & Coal Preparation		
Plants $150,000	150,000	
Investment Total	1,964,500	
B. Amortization 5%/year		
5% x 1,964,500 ÷ 350 days	281.00	.54

III CAPITAL COSTS (Continued)	per day	per ton coke
C. Interest 5% on unpaid balance		
1/2 time x 5.0% x \$1,964,500		
÷ 350 days	141.00	.27
Capital Cost Total	422.00	.81
TOTAL COSTS		
I RAW MATERIALS	4,752.00	9.22
II OPERATING COSTS	965.04	1.89
III CAPITAL COSTS	422.00	.81
TOTAL COSTS	6,139.04	11.92
IV INCOME, BY-PRODUCT		
A. Tar oil .06/gal; 90 tons/day		
.06/gal. x 90 tons x 2000 lbs./ton		
÷ 8 lbs./gal.	1,350.00	2.62
B. Surplus Gas - Heat Value		
3168 x 10^6 Btu/day		
Heat Value - Coal		
22.3 x 10^6 Btu/ton coal		
$\frac{5.40/\text{ton} \times 3168 \times 10^6 \text{ Btu}}{22.3 \times 10^6 \text{ Btu/ton}}$	766.80	1.39
Total Income (By-Product)	2,116.80	4.01

V	NET COKE COSTS	per day	per ton coke
	TOTAL PLANT COSTS	6,139.04	11.92
	LESS INCOME	2,116.80	4.01
	NET COKE COSTS	4,022.24	7.91

C. Krupp Renn Process Costs For Nickeliferous Ore

Basic Variable Data	Assumptions
a. Luppen production	
$\frac{\%\text{ Fe in product}}{\%\text{ recovery}} \times \frac{100}{\%\text{ Fe in ore}} = \frac{\text{No. tons ore}}{\text{Ton luppen}}$	
$\frac{91\%}{92\%} \times \frac{100\%}{19.5\%} =$	5.07
b. Cost of ore per ton at mine - dried	2.00
c. Freight on ore per ton - mine to plant	.50
d. Cost of flux per ton at source	4.00
e. Freight on flux per ton - source to site	1.00
f. Cost of coal per ton at mine	5.00
g. Freight on coal per ton - mine to site	.40
h. Cost of coke per ton at source	7.91
i. Freight on coke per ton - source to site	4.25
j. Cost of water at site - cents per 1000 gallons	.25
k. Cost of electric power at site - cents per kwh	.01
l. Cost of labor - hourly rate	2.30
m. Amount of fuel for reduction expressed in % per ton ore	28%
n. Amount of fuel for heating expressed in % per ton ore	8%
o. Amount of flux required expressed in % per ton ore	5%

Basic Fixed Data

Amortization - 5%/year on investment
Interest - 5%/year on 1/2 investment
Maintenance - 4%/year on investment
Labor - Hourly rate + 20% fringe benefits
Salaries & Supervision - 20% of wages

Raw Material costs per ton of luppen

Iron ore needs

Cost per ton of ore	2.50/ton ore
Cost per ton luppen	
5.07 tons ore/ton luppen x 2.50 =	12.68/ton luppen

Flux needs

Flux 5%	
Cost	\$ 5.00/ton
Cost per ton of ore	
5% x \$5.00/ton	\$ .25/ton
Cost per ton of luppen	
5.07 x cost per ton of ore \$ .25/ton	\$ 1.27/ton

Fuel needs

Reduction fuel	28%/ton ore	
5.07 x 28%		142%/ton luppen
Heating fuel	8%/ton ore	
5.07 x 8%		40.6%/ton luppen
Cost - Coal 5.00 + .40		\$5.40/ton
Coke 7.91 + 4.25		\$12.16/ton
Cost of coke per ton ore		
$\frac{28}{100} \times 12.16$		\$3.40
Cost of coke per ton luppen		
$5.07 \times \frac{28}{100} \times 12.16$		\$17.99

Cost of coal per ton ore

$\frac{8}{100} \times 5.40 =$	.43

Cost of coal per ton luppen

$5.07 \times \frac{8}{100} \times 5.40$	2.18

<u>Total raw material costs per ton luppen</u>

1.	Ore		12,68
2.	Flux		1.27
3.	Fuel - reduction coke		17.24
	heating coal		2.18
		TOTAL	33.37

Operating Cost Per Ton Of Luppen

Plant Size	3 Kilns of 15 x 360 feet
Investment Cost	$ 15, 000, 000
Production of luppen per year	119, 000 tons
Ore throughput per day & per kiln	616 tons

Power - 50 kwh/ton dry ore	
5. 07 tons ore/ton luppen x 50 kwh x . 01	2. 54
Heating of kiln - 10°/hr. to 720° after relining	
2. 5 tons coal/hr. x $\frac{720^{\circ}}{10^{\circ}/\text{hr.}}$ x 5. 40 x	
2 times/year ÷ 108, 000 tons luppen	. 02
Maintenance & Repair	
$\frac{4\%}{100}$ x $\frac{15,000,000 \text{ investment}}{119,000 \text{ tons luppen}}$	5. 04
Kiln Lining	
$\frac{171,550}{119,000}$ tons luppen	1. 44
Slag Disposal	
This covers movement to & disposal on slag pile	. 10
Water - 350 gals/ton luppen for quenching & cooling	
Rate - . 25/1000 gals x 350 gals.	. 09

Wages *

76 men x 8 hrs. x 325 days = 197,600 man hrs.

$\frac{197{,}600 \text{ man hrs.}}{119{,}000 \text{ tons luppen}}$ = 1.70 man hrs. per ton luppen

$\frac{\$509{,}195 \text{ Oper. Labor Cost}}{197{,}600 \text{ man hrs.}}$ = 2.58 cost/man hr.

1.70 x 2.58 = $ 4.39

Overhead

This is equal to 20% of the basic wage rate

$\frac{20\%}{100}$ x 4.39 = .88

Total Operating Cost Per ton Luppen $ 14.50

Capital Costs Per Ton Luppen

Investment - $15,000,000

Interest - 5% on 1/2 investment

1/2 x $\frac{5\%}{100}$ x 15,000,000 ÷ 119,000 tons luppen 3.15

Amortization

$\frac{5\%}{100}$ x 15,000,000 ÷ 119,000 tons luppen 6.30

$ 9.45

* See the following pages

ESTIMATED FORCE REQUIREMENT

KRUPP - RENN PROCESS FOR IRON PRODUCTION USING THREE ROTARY KILNS

Occupation	Force: Per Turn 1 2 3	Force: Per Day Total	Hourly (Or monthly) rate	cost per day (2)
General				
General - Foreman	0-1-0	1	900	29.60
Asst. Genl. Foreman	1-1-1	3	800	78.95
Chief Clerk	0-1-0	1	650	21.38
Clerks	0-1-1	2	2,150	34.83
Chemical Analyst	0-1-0	1	650	21.38
Sampleman	1-1-1	3	2,215	54.46
Warehouseman	0-1-0	1	2,280	18.24
Janitor	0-1-0	1	1,190	15.12
Total	2-8-3	13		273.86
Storage Yard & Crusher				
Foreman - Turn	0-1-1	2	650	42.76
Crusher Operator	0-1-1	2	2,345	38.00
Coal Pulverizer Oper.	0-1-1	2	2,150	34.83
Craneman	0-1-1	2	2,280	36.86
Yardman	0-1-1	2	2,085	33.84
Conveyorman	0-1-1	2	2,085	33.84
Greaser	0-1-1	2	2,085	33.84
Total	0-9-9	14		254.07
Kilns				
Foreman - Turn	1-1-1	3	700	69.08
Kiln Operator	2-2-2	6	2,475	121.20
Kiln Operator helper	2-2-2	6	2,215	108.72
Feeder Attendant	2-2-2	6	2,150	105.60
Fuel Coal Fan Tender	2-2-2	6	2,150	105.60
Conveyorman	1-1-1	3	2,085	51.24
Greaser	1-1-1	3	2,085	51.24
Total	11-11-11	33		612.68
Milling & Separation				
Foreman - Turn	0-1-1	2	650	42.76
Milling Plant Oper.	0-1-1	2	2,345	38.00
Magnetic Sep. Oper.	0-1-1	2	2,345	38.00
Craneman	0-1-1	2	2,280	36.86
Car Loader	0-1-1	2	2,150	34.88
Conveyorman	0-1-1	2	2,085	33.84
Greaser	0-1-1	2	2,085	33.84
Total	0-7-7	14		258.28

Maintenance	Kiln reline			
Foreman Maintenance				
Foreman - Turn	1-1-1	3	650	64.14
Repairman-Crushers				
Welder - Crushers				
Repairman - Kilns	1-1-1	3	2,670	65.28
Repairman - Mil'g	1-1-1	3	2,670	65.28
Repairman - Electrical				
Bricklayer	1-2-1	4	2,605	84.56
Carpenter				
Machinist - Repair Shop				
Welder - Repair Shop				
Laboror	1-2-1	4	1,890	61.68
Total		17		340.94

Maintenance	General			
Foreman - Maintenance	0-1-1	1	750	24.67
Foreman - Turn	1-1-1	3	650	64.14
Repairman - Crushers	0-2-2	4	2,670	86.40
Welder - Crushers	0-1-1	2	2,670	43.20
Repairman - Kilns	2-2-2	6	2,670	129.60
Welder - Kilns	1-1-1	3	2,670	65.28
Repairman Mil'g & Sep.	0-1-1	2	2,670	43.20
Repairman - Electrical	1-2-2	5	2,670	108.60
Bricklayer	1-2-1	4	2,605	84.56
Carpenter	1-1-1	3	2,605	63.72
Machinist - Repair Shop	0-2-0	2	2,670	42.72
Welder - Repair Shop	0-1-0	1	2,670	21.36
Laboror	2-4-2	8	1,890	123.26
Total	9-21-14	44		900.81

Operating Labor Cost

General	$ 273.86
Storage Yard & Crusher	254.07
Kilns	612.68
Milling & Separation	258.28
	$1,398.89

$1,398.89 x 325 days = $454,639 per year

$454,639 x 12% fringe benefits = total cost/year $ 509,195

Reline of Kiln

Material Cost

Reline 2/3 of kiln each 30 months

2/3 length ÷ 5/2 year = .266 length/year

Reline 1/3 of kiln each 8 months

1/3 length ÷ 2/3 years = .5 length/year

Brick cost per = $42,113

Brick cost per year = 3 kilns x $42,113 x .766

= $96,858/year 56.5%

Labor Cost

Reline labor = $340.94/day

($340.94 + 12% fringe benefits plus
20% overhead) x 163 days = $74,692/year 43.5%

100.0%

$ 171,550

Krupp - Renn Plant

Total Cost Per Ton Luppen

Plant Size	3 kilns of 15 x 360 feet
Investment Cost	$15,000,000
Raw Materials	33.37
Operating Costs	14.50
Capital Costs	9.45
Total Costs Per Ton Luppen	57.32

D. PROFIT POTENTIAL

The profit situation of a plant of this nature would vary according to the market for nickel. The value of the iron is considered to be $.03/pound.

Price of Nickel			Profit Per Ton Luppen	119,000 tons/year Profit per year
Ni	$ .74/#	116.76 - 57.32	$ 59.44	$ 7,070,000
Ni	$1.42/#	173.88 - 57.32	$116.56	$ 13,880,000
Ni	$2.00/#	222.60 - 57.32	$165.28	$ 19,700,000

This does not consider cost of sales, freight to customer or corporation overhead.

In examining this potential the assumptions concerning the ore, the process and the limitations of the market analysis must be borne in mind.

Legends

		Page No.
Fig. 1.	Map indicating Gold Beach area of Oregon	3
Fig. 2.	Red Flats Nickel Deposit	20
Fig. 3.	Color aerial photograph	27
Fig. 4.	Red Flats Deposit Plot Plan	26
Fig. 5.	Map of Gold Beach Quandrangle	(in back pocket)

RED FLAT PLACERS

Near Gold Beach, Oregon.

Location

The Red Flat Placers are located in Sections 18, 19 and 30, Township 37 S., Range 13 W. and Sections 13 and 24, Township 37 S., Range 14 W., about eight miles air-line southeast of Gold Beach. The property can be reached by going south from Gold Beach to Pistol River then taking Pyramid Rock Lookout road. Sixteen miles of grade in which a gain of approximately 2000 feet in elevation is made. Barometer elevation of property is 3500 feet.

Property.

The property consists of nine placer claims of one hundred sixty acres each and known as the Red Gold Association No. 1 to 9 inclusive. See attached sketch for relative location.

Facilities.

There is no timber on the claims, but plenty of timber available a few miles to the west of the property. There are a number of small springs on the Flat for domestic water, and plenty of water available in Hunter Creek and Pistol River for commercial use.

Geology.

The Red Flat Placers is just what the name implies. Upon weathering, it has become a bright red color indicating a large iron oxide content. The product of erosion on either side is not red in color so a fairly distinct boundary line is formed. The country rock is serpentine and there appears to be no distinctive difference in the serpentine rocks to the west or east of the deposit. So far as could be determined there were no rocks in place. The Red Flat material has weathered much faster than the adjoining serpentine, and formed alluvial deposit. The terrain being fairly level the product of erosion is not carried away. The deepest prospect hole is thirty-two feet deep and it is said to be in the eroded material (now caved in).

Sampling and Assaying.

Mr. Smedberg informed me that fire assays would be of no avail and the values to be recovered would have to be extracted by acid. I had Mr. Smedberg point out a good place to take samples. I took a sample from this place by digging a hole 3 feet deep into the eroded material which was loose and a hole could be dug with a shovel. There being a few igneous rocks loose in the material, the igneous rocks were saved and made sample #1 which run 70¢. Sample #2 was the red material taken from top to bottom of the hole. This sample showed no value either in gold or silver.

Several hand specimens were taken of the serpentine and after returning home it was decided to see if the serpentine contained any values. These were made into sample #3 and found to contain no value in gold or silver.

Those interested in the Red Flat Placers have their assays run by the wet method (and ~~attached hereto are two copies~~) by Paul Smith Laboratories of 9315 Pico Blvd Los Angeles of July 18th and September 9, 1933. Mr. Smedberg expressed the opinion that these two assays are very close to the general average of the deposit.

These wet assays claim $20.00 gold, [illegible] silver, $20.00 nickel, and $20.00 in platinum and allied metals

Conclusions.

As the deposit is enormous in size, not less than 1500 acres, with an average of say three feet in depth, the question becomes values and not quantity. The value as far as gold and silver are concerned is doubtful. If rare minerals are present they are unknown. A number of metals are present but none commercial at present time. I believe the owners are fooling themselves by putting faith into unorthodox analysis.

October 4, 1937.

(signed) J. E. Morrison

J. E. Morrison,
Mining Geologist.

(C O P Y)

P A U L   S M I T H   L A B O R A T O R I E S .

9315 W. Pico Blvd., Los Angeles, California

September 9, 1933.

RESEARCH ASSAY ON THE RED IRON ORE OF RED FLATS, OREGON. This ore is a concentration and alteration product of the above mentioned Nickel-bearing serpentine.

	Took for assay	50 grms.	or	100% ore	
1.	On roasting I lost in the form of moisture, organic matter, arsenic, etc.	5.20 gr.	or	10.40%	"
2.	Leached the ore with acid, found heavy concentrate,	3.00 "	"	6.00%	"
3.	Found coarse insoluble quartz, iron stained, etc.	6.15 "	"	12.30%	"
4.	Red Oxide of Iron and Chrome	8.20 "	"	16.40%	"
5.	Found very heavy concentrates and undissolved matter	1.00 "	"	2.00%	"
	TOTAL RECLAIMING	23.55 "	"	47.10%	"
6.	In acid we have	26.45 "	"	52.90%	"
		50.00 "	"	100.00%	"

ACCOUNT OF ACID SOLUBLE MATTER.

7.	Found Ferric oxide	22.65 "	"	45.30%	"
	Substance dissolved by acid	26.45 "	"	52.90%	"
		49.10 "	"	98.20%	"
	Balance of metals in solution	.90 "	"	1.80%	"

8. Nickel in acid solution 1.12% as the oxide.

Actual recovery of Nickel from 50 grms. was 0.385 grms., or 0.385 x 2 equals .770% Nickel oxide.
(4) gave additional Nickel; 0.14 x 16.40 equals 0.33% NiO

(2) gave 45% Chromium and only a trace of Nickel (0.02)
Total Nickel recovered is 1.10% NiO or 0.86% Nickel.

Signed D. Petcoff

Chemist.

(C O P Y)

Los Angeles, California, July 18, 1933.

Dear Col. Johnson:-

Pursuant to contract I submit the following laboratory report, which I certify as correct.

(Signed) Paul Smith.

To

Mr. PAUL SMITH:

As per your request I have compiled from my notebook, data which will indicate the extractable values in samples known as Oregon Red Flats. These are chemical assays and indicate only what can be taken out by your process and not extractable by orthodox methods.

GOLD:

1. Aqua Regia 1 A. T. 1 mg. button, or 1 oz per ton
2. Chlorination 1 A.T.1 mg. " " 1 " " "
3. Bromine 1 A.T. 1 mg " " 1 " " "
4. Chlorine fire assay 1/10 A.T. 3/4 mg button over, 20% gold or platinum or 1.5 oz per ton.

PLATINUM AND ALLIED METALS.

1. Mort flux platinum sponge 1/3 A.T. 0.43 mg. button, or 1.3 oz per ton.
2. Chlorination tails .2 A.T. platinum sponge .26 mg. button 1.3 oz. per ton
3. Inquarted silver button parted 1 A.T. yellow liquid indicated over .01 mg. of palladium. Trace of Osmium in residue.

SILVER:

1. Sodium Thyo Sulphate 20 gr. 10 mg. button, or 15 oz. pr.ton.

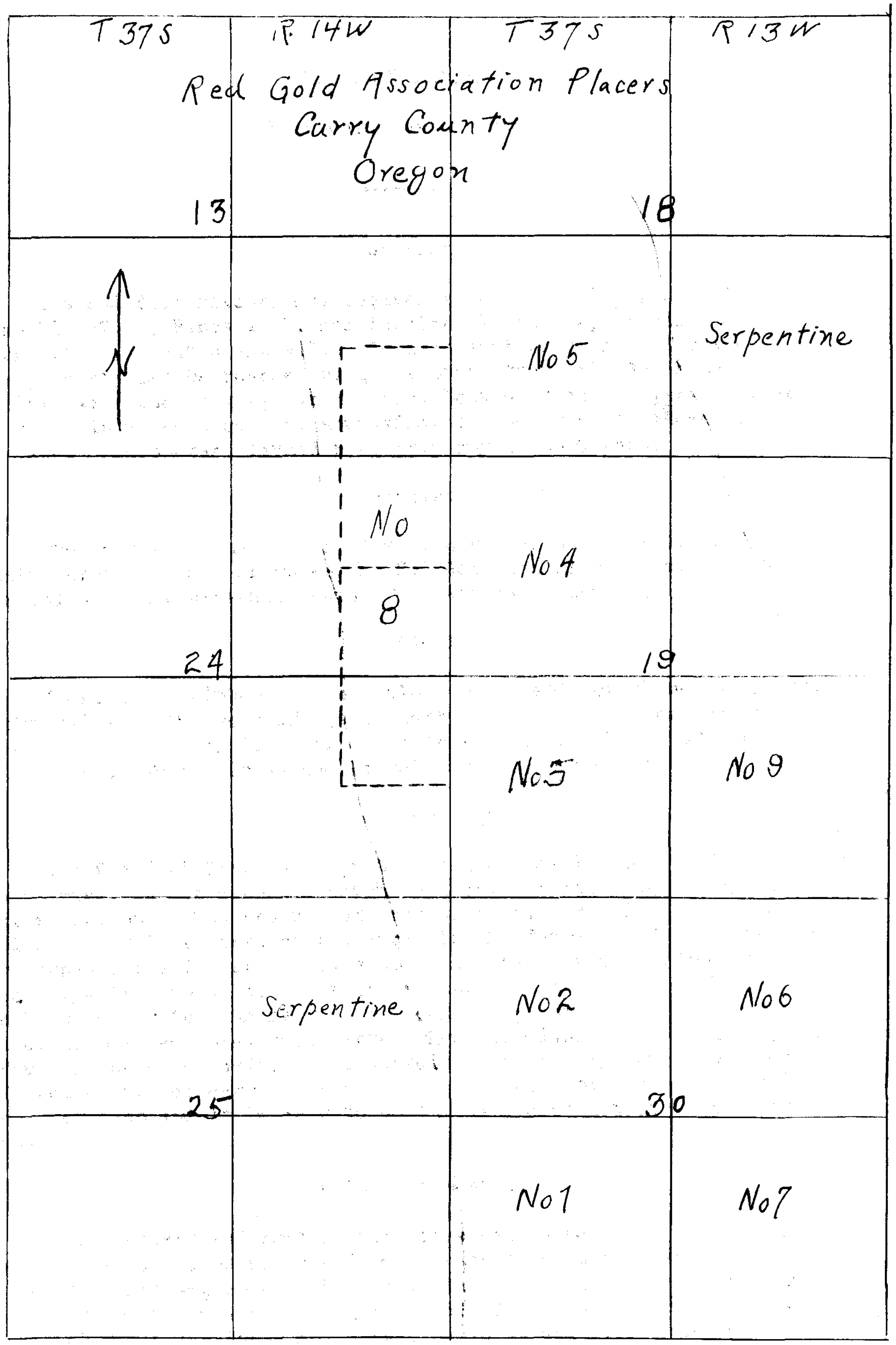
NICKEL:

1. 1.3% NICKEL, acid soluble, both by chemical and electrical precipitations.
2. Insoluble NICKEL has not been determined but indications are another 1% in residue.

At the present market quotations, it is conservative to say, this ore contains twenty dollars in gold, twenty dollars in Platinum and allied metals, twenty dollars in Nickel and six dollars in Silver.

The Mercury has not been satisfactorily determined, but indications are at least four dollars in Mercury.

(SIGNED) Robert Boyer, Chemist,
Metals Reduction Company,
9315 Pico Blvd., Los Angeles, Calif.



T 37 S
R 14 W
T 37 S
R 13 W
Red Gold Association Placers
Carry County
Oregon
13
18
N
No 5
Serpentine
No
8
No 4
24
19
No 5
No 9
Serpentine
No 2
No 6
25
30
No 1
No 7

Motz

RECEIVED OCT 1 1940 STATE DEP'T OF GEOLOGY & MINERAL INDS.

STATE DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES

ASSAY REPORT

Office Number AG-578 AG-579 AG-580

Grants Pass, Oregon
B~~xxxxxxxxxx~~

September 30, 19~~3~~ 40

Sample submitted by M. T. Edwards, Burnett Motor Co., Portland, Oregon

Sample description AP-12A--Black sand containing magnetite, hematite, & chromite. AP-12B--Sand, principal constituents are magnetite and rock fragments. AP-12C--Altered rock containing a large amount of limonite.

The assay results given below are made without charge as provided by Chapter 176, Section 10, Oregon Laws 1937, the sender having complied with the provisions thereof.

NOTICE: The assay results given below are from a sample furnished by the above named person. This department had no part in the taking of the sample and assumes no responsibility, other than the accuracy of the assay of the material as furnished it by the sender.

Sample Number	GOLD		MERCURY ~~xxxxxx~~		CHROMIC OXIDE		NICKEL		
	Ounces per ton	Value	Ounces per ton	Value	Percent	Value	Percent	Value	Total Value
AP-12-A			0.055		40.9		Blank		
AP-12-B			Blank				Blank		
AP-12-C			Blank				Blank		

Market Quotations:
Gold $ per oz.
Silver $ per oz.
$ per oz.
$ per oz.

STATE ASSAY LABORATORY

Albert A. Lewis
Assayer

Red Flat Placers
Gold Beach Dist

A+8215

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 1 Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar 2340' Aneroid Should be corrected					
Depth to top of bed	0'0"	1'2"	1'2"	1	deep reddish brown soil,
Elev. of top of bed					weathered peridotite
Thickness of bed	1'2"	2'0"	10"	2	soft yellow soil
Elev. bottom bed	2'0"	3'0"	1'0"	3	begins to be gritty at 2',
Depth of hole 11 ft.					less weathered - yellow, piece
Elev. water table not encountered					of small boulder at 2'6"
Bottomed in peridotite rock	3'0"	4'0"	1'0"	4	yellow lumpy soil - soft
***********	4'0"	5'0"	1'0"	5	yellow,clayey, very soft
Drill used auger & chopping					greenish black streaks
Number men 3	4'6"	5'0"	0'6"	5A	moisture sample taken
Engr. in charge Libbey					Jar no. 27
Mtrl. classfd. by	5'0"	6'0"	1'0"	6	yellow to greenish black, very
Sampler Libbey					soft, moist, clayey, plastic
Date hole began May 31, 1946	6'0"	7'0"	1'0"	7	same as no. 6
Date hole finished May 31, 1946	7'0"	8'0"	1'0"	8	moister, more even colored, dark streaks
Shifts actually drilled 3 P.M. to 4:30 P.M.	7'0"	8'0"	1'0"	8A	Jar 30, same as 8

(Continued on next page)

Remarks 38.19 (Mile post 14) 38.47 Jct. with road to mining camp.

On June 3 the altimeter at Brookings (a little above sea level) read minus 150 feet.

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 1-Cont. Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

Elev. collar

Depth to top of bed

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole

Elev. water table

Bottomed in peridotite rock

****

Drill used

Number men

Engr. in charge

Mtrl. classfd. by

Sampler

Date hole began

Date hole finished

Shifts actually drilled

From	To	Thickness of sample	Sample No.	Description
8'0"	9'0"	1'0"	9	moister, still yellow, brown
				spots, clayey, very plastic and
				wet, occasional lumps
9'0"	10'0"	1'0"	10	same as 9 with partly weathered
				peridotite pebble, muddy
10'0"	11'0"	1'0"	11	Jar no. 32, muddy, numerous
				small partially weathered
				peridotite lumps, greenish
				lumps, still yellow soil
				Encountered rock at 11 ft.
				where hole was bottomed.

Remarks

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. ______ Location 2600 ft. Apparently should be 2600 feet (45 paces - 5.8 ft. to pace) (Mr. Libbey's notes say 500 ft. 544' W. of pt. on road. S. 45°W. to 50°W. of point on road. 0.19 mi. S. of hole no. 1 and 0.06 mi. N. of milepost 14. (N 25 E. from hole no. 2 to Pyramid Rock) 200 ft. E. of high point or ridge top of flat where rock crops out-about 30 ft. higher

of sec. ______ T. ______ R. ______ County Curry

Coord. ______ N. ______ E. ______

Elev. collar Aneroid 2360' Check before leaving gave 2370'
Depth to top of bed ______
Elev. of top of bed ______
Thickness of bed ______
Elev. bottom bed ______
Depth of hole 7 feet
Elev. water table not encountered
Bottomed in peridotite rock

****

Drill used auger, chopping
Number men 3
Engr. in charge Libbey
Mtrl. classfd. by "
Sampler "
Date hole began June 1, 1946
Date hole finished June 1, 1946
Shifts actually drilled 1 hour

From	To	Thickness of sample	Sample No.	Description
0'0"	1'0"	1'0"	13	brown with dk. gray streaks
1'0"	2'0"	1'0"	14	brown dark moist magnetic soil
2'0"	3'0"	1'0"	15	moister, brown, serpentine chunk
3'0"	4'0"	1'0"	16	very moist, yellowish gray
4'0"	5'0"	1'0"	17	very moist, yellowish gray,
				quite plastic, Jar no. 38,
				varicolored
5'0"	6'0"	1'0"	18	very moist, yellowish gray
6'0"	7'0"	1'0"	19	very moist, gritty, reddish
				brown globs and streaks, still
				gray bottomed at 7'0" - hit
				peridotite rock.

Remarks Sample no. 12 is sack of shot collected 30 ft. west of road. 0.19 mi. S. of hole 1.
Drilled 2 ft. at site and moved over to south 3 ft. and picked up sampling at 2 feet.
Sample no. 20 is rock exposed at hole 2 and sample no. 21 is "shot" collected on surface at hole 2.

WL-24b

Hole No. 2

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 3 Location Roadcut on west side of Pistol R. road where others have sampled 0.20 mi. south of milepost 14 and 0.45 mi. south of hole no. 1

of sec. ______ T. ______ R. ______ County Curry

Coord. ______ N. ______ E. ______

Elev. collar 2250' which is 1 ft. above road level

Depth to top of bed ______

Elev. of top of bed ______

Thickness of bed ______

Elev. bottom bed ______

Depth of hole 7 feet

Elev. water table ______

Bottomed in peridotite rock

****

Drill used auger and chopping

Number men 3

Engr. in charge Libbey

Mtrl. classfd. by "

Sampler "

Date hole began June 1, 1946

Date hole finished ______

Shifts actually drilled 1 hour

From	To	Thickness of sample	Sample No.	Description
			22	2 ft. channel sample of yellowish
				brown soil, random rock in bank
				above collar of hole
0'0"	1'0"	1'0"	23	soft yellowish brown-firm
1'0"	2'0"	1'0"	24	" " " "
2'0"	3'0"	1'0"	25	" " " "
3'0"	4'0"	1'0"	26	Jar no. 31 ", still no lumps
4'0"	5'0"	1'0"	27	yellow brown with green black spots
5'0"	6'0"	1'0"	28	darker-brown-moist " " spots
6'0"	7'0"	1'0"	29	" " moister" " "
				bottomed at 7'0" - peridotite
				rock encountered

Remarks sample no. 22 is overlain by 1 foot of deep brown soil. Mining camp Jct. is 0.55 mi. s. of hole no. 1 and 0.1 mi. s. of hole no. 3. Sample no. 30 is peridotite rock with relatively large magnetite grains. Occurs as float at end of road leading n.w. and down from Jct. with Pistol R. Road and Mining Camp Road; about 1 mile. Sample no. 31 is of

WL-24b

Hole No. 3

(over)

greenstone(?) which occurs as isolated masses in peridotite areas, occurs on road to N.W. in several spots. Impression is that they are roof pendants. The rock is much jointed with fillings in them. Schist containing quartz occurs on hill south of saddle of junction of 3 roads. It is probably a part of the Colebrooke fm. Sample no. 32 is of greenstone N. E. of hole no. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Location on lower flat 0.10 mi. S. 67°E. of house - sho at camp

of sec. T. R. County Curry

N. E.

" aneroid corrected	From	To	Thickness of sample	Sample No.	Description
	0'0"	1'0"	1'0"	33	chocolate brown soil
	1'0"	2'0"	1'0"	34	lighter brown, loose rocks
	2'0"	3'0"	1'0"	35	brown rocky with white pieces
	3'0"	4'0"	1'0"	36	brown, very rocky (peridotite)
	4'0"	5'0"	1'0"	37	brownish gray, clayey
	5'0"	6'0"	1'0"	38	brown and gray, clayey
	6'0"	7'0"	1'0"	39	moist gray, clayey, Jar 26
***	7'0"	7'10	10 in.	40	bottomed at 7'10' - hit
					peridotite rock, moist, gray
					and clayey
Libbey					
"					
"					
ne 1, 1946					
June 1, 1946					
lled					

Hole No. 4

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 1 Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar 2340' Aneroid Should be corrected					
Depth to top of bed	0'0"	1'2"	1'2"	1	deep reddish brown soil,
Elev. of top of bed					weathered peridotite
Thickness of bed	1'2"	2'0"	10"	2	soft yellow soil
Elev. bottom bed	2'0"	3'0"	1'0"	3	begins to be gritty at 2',
Depth of hole 11 ft.					less weathered - yellow, piece
Elev. water table not encountered					of small boulder at 2'6"
Bottomed in peridotite rock	3'0"	4'0"	1'0"	4	yellow lumpy soil - soft
***********	4'0"	5'0"	1'0"	5	yellow, clayey, very soft
Drill used auger & chopping					greenish black streaks
Number men 3	4'6"	5'0"	0'6"	5A	moisture sample taken
Engr. in charge Libbey					Jar no. 27
Mtrl. classfd. by	5'0"	6'0"	1'0"	6	yellow to greenish black, very
Sampler Libbey					soft, moist, clayey, plastic
Date hole began May 31, 1946	6'0"	7'0"	1'0"	7	same as no. 6
Date hole finished May 31, 1946	7'0"	8'0"	1'0"	8	moister, more even colored, dark streaks
Shifts actually drilled 3 P.M. to 4:30 P.M.	7'0"	8'0"	1'0"	8A	Jar 30, same as 8

(Continued on next page)

Remarks 38.19 (Mile post 14) 38.47 Jct. with road to mining camp.

On June 3 the altimeter at Brookings (a little above sea level) read minus 150 feet.

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 1-Cont. Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

Elev. collar
Depth to top of bed
Elev. of top of bed
Thickness of bed
Elev. bottom bed
Depth of hole
Elev. water table
Bottomed in peridotite rock

****

Drill used
Number men
Engr. in charge
Mtrl. classfd. by
Sampler
Date hole began
Date hole finished
Shifts actually drilled

From	To	Thickness of sample	Sample No.	Description
8'0"	9'0"	1'0"	9	moister, still yellow, brown
				spots, clayey, very plastic and
				wet, occasional lumps
9'0"	10'0"	1'0"	10	same as 9 with partly weathered
				peridotite pebble, muddy
10'0"	11'0"	1'0"	11	Jar no. 32, muddy, numerous
				small partially weathered
				peridotite lumps, greenish
				lumps, still yellow soil
				Encountered rock at 11 ft.
				where hole was bottomed.

Remarks

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. ______ Location 2600 ft. Apparently should be 2600 feet (45 paces - 5.8 ft. to pace) (Mr. Libbey's notes say 500 ft. 544' W. of pt. on road. S. 45°W. to ~~50°W. of point on road. 0.19 mi. S. of hole no. 1 and 0.06 mi. N. of~~ milepost 14. (N 25 E. from hole no. 2 to Pyramid Rock) 200 ft. E. of high point or ridge top of flat where rock crops out-about 30 ft. higher

_____ of sec. _____ T. _____ R. _____ County Curry

Coord. _____ N. _____ E. _____

Elev. collar Aneroid 2360' Check before leaving

Depth to top of bed gave 2370'

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole 7 feet

Elev. water table not encountered

Bottomed in peridotite rock

****

Drill used auger, chopping

Number men 3

Engr. in charge Libbey

Mtrl. classfd. by "

Sampler "

Date hole began June 1, 1946

Date hole finished June 1, 1946

Shifts actually drilled 1 hour

From	To	Thickness of sample	Sample No.	Description
0'0"	1'0"	1'0"	13	brown with dk. gray streaks
1'0"	2'0"	1'0"	14	brown dark moist magnetic soil
2'0"	3'0"	1'0"	15	moister, brown, serpentine chunk
3'0"	4'0"	1'0"	16	very moist, yellowish gray
4'0"	5'0"	1'0"	17	very moist, yellowish gray,
				quite plastic, Jar no. 38,
				varicolored
5'0"	6'0"	1'0"	18	very moist, yellowish gray
6'0"	7'0"	1'0"	19	very moist, gritty, reddish
				brown globs and streaks, still
				gray bottomed at 7'0" - hit
				peridotite rock.

Remarks Sample no. 12 is sack of shot collected 30 ft. west of road. 0.19 mi. S. of hole 1. Drilled 2 ft. at site and moved over to south 3 ft. and picked up sampling at 2 feet. Sample no. 20 is rock exposed at hole 2 and sample no. 21 is "shot" collected on surface at hole 2.

WL-24b

Hole No. 2

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 3 Location Roadcut on west side of Pistol R. road where others have sampled 0.20 mi. south of milepost 14 and 0.45 mi. south of hole no. 1

of sec. T. R. County Curry

Coord. N. E.

Elev. collar 2250' which is 1 ft. above road level

Depth to top of bed

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole 7 feet

Elev. water table

Bottomed in peridotite rock

****

Drill used auger and chopping

Number men 3

Engr. in charge Libbey

Mtrl. classfd. by "

Sampler "

Date hole began June 1, 1946

Date hole finished

Shifts actually drilled 1 hour

From	To	Thickness of sample	Sample No.	Description
			22	2 ft. channel sample of yellowish
				brown soil, random rock in bank
				above collar of hole
0'0"	1'0"	1'0"	23.	soft yellowish brown-firm
1'0"	2'0"	1'0"	24	" " " "
2'0"	3'0"	1'0"	25	" " " "
3'0"	4'0"	1'0"	26	Jar no. 31 ", still no lumps
4'0"	5'0"	1'0"	27	yellow brown with green black spots
5'0"	6'0"	1'0"	28	darker-brown-moist " " spots
6'0"	7'0"	1'0"	29	" " moister" " "
				bottomed at 7'0" - peridotite
				rock encountered

Remarks sample no. 22 is overlain by 1 foot of deep brown soil. Mining camp Jct. is 0.55 mi. s. of hole no. 1 and 0.1 mi. s. of hole no. 3. Sample no. 30 is peridotite rock with relatively large magnetite grains. Occurs as float at end of road leading n.w. and down from Jct. with Pistol R. Road and Mining Camp Road; about 1 mile. Sample no. 31 is of

WL-24b

Hole No. 3

(over)

greenstone(?) which occurs as isolated masses in peridotite areas, occurs on road to N.W. in several spots. Impression is that they are roof pendants. The rock is much jointed with fillings in them. Schist containing quartz occurs on hill south of saddle of junction of 3 roads. It is probably a part of the Colebrooke fm. Sample no. 32 is of greenstone N. E. of hole no. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 4 Location on lower flat 0.10 mi. S. 67°E. of house - shop at camp

of sec. T. R. County Curry

Coord. N. E.

Elev. collar 2250' aneroid
Should be corrected
Depth to top of bed
Elev. of top of bed
Thickness of bed
Elev. bottom bed
Depth of hole
Elev. water table
Bottomed in

****

Drill used
Number men
Engr. in charge Libbey
Mtrl. classfd. by "
Sampler "
Date hole began June 1, 1946
Date hole finished June 1, 1946
Shifts actually drilled

From	To	Thickness of sample	Sample No.	Description
0'0"	1'0"	1'0"	33	chocolate brown soil
1'0"	2'0"	1'0"	34	lighter brown, loose rocks
2'0"	3'0"	1'0"	35	brown rocky with white pieces
3'0"	4'0"	1'0"	36	brown, very rocky (peridotite)
4'0"	5'0"	1'0"	37	brownish gray, clayey
5'0"	6'0"	1'0"	38	brown and gray, clayey
6'0"	7'0"	1'0"	39	moist gray, clayey, Jar 26
7'0"	7'10	10 in.	40	bottomed at 7'10' - hit
				peridotite rock, moist, gray
				and clayey

Remarks

WL-24b

Hole No. 4

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 1 Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

Elev. collar 2340' Aneroid Should be corrected
Depth to top of bed
Elev. of top of bed
Thickness of bed
Elev. bottom bed
Depth of hole 11 ft.
Elev. water table not encountered
Bottomed in peridotite rock
****
Drill used auger & chopping
Number men 3
Engr. in charge Libbey
Mtrl. classfd. by
Sampler Libbey
Date hole began May 31, 1946
Date hole finished May 31, 1946
Shifts actually drilled 3 P.M. to 4:30 P.M.

From	To	Thickness of sample	Sample No.	Description
0'0"	1'2"	1'2"	1	deep reddish brown soil,
				weathered peridotite
1'2"	2'0"	10"	2	soft yellow soil
2'0"	3'0"	1'0"	3	begins to be gritty at 2',
				less weathered - yellow, piece
				of small boulder at 2'6"
3'0"	4'0"	1'0"	4	yellow lumpy soil - soft
4'0"	5'0"	1'0"	5	yellow,clayey, very soft
				greenish black streaks
4'6"	5'0"	0'6"	5A	moisture sample taken
				Jar no. 27
5'0"	6'0"	1'0"	6	yellow to greenish black, very
				soft, moist, clayey, plastic
6'0"	7'0"	1'0"	7	same as no. 6
7'0"	8'0"	1'0"	8	moister, more even colored, dark streaks
7'0"	8'0"	1'0"	8A	Jar 30, same as 8

(Continued on next page)

Remarks 38.19 (Mile post 14) 38.47 Jct. with road to mining camp.

On June 3 the altimeter at Brookings (a little above sea level) read minus 150 feet.

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 1-Cont. Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar					
Depth to top of bed	8'0"	9'0"	1'0"	9	moister, still yellow, brown
Elev. of top of bed					spots, clayey, very plastic and
Thickness of bed					wet, occasional lumps
Elev. bottom bed	9'0"	10'0"	1'0"	10	same as 9 with partly weathered
Depth of hole					peridotite pebble, muddy
Elev. water table	10'0"	11'0"	1'0"	11	Jar no. 32, muddy, numerous
Bottomed in peridotite rock					small partially weathered
***********					peridotite lumps, greenish
Drill used					lumps, still yellow soil
Number men					Encountered rock at 11 ft.
Engr. in charge					where hole was bottomed.
Mtrl. classfd. by					
Sampler					
Date hole began					
Date hole finished					
Shifts actually drilled					

Remarks

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. ______ Location 2600 ft. Apparently should be 2600 feet (45 paces - 5.8 ft. to pace) (Mr. Libbey's notes say 500 ft. 544' W. of pt. on road. S. 45°W. to 50°W. of point on road. 0.19 mi. S. of hole no. 1 and 0.06 mi. N. of milepost 14. (N 25 E. from hole no. 2 to Pyramid Rock) 200 ft. E. of high point or ridge top of flat where rock crops out-about 30 ft. higher

______ of sec. ______ T. ______ R. ______ County Curry

Coord. ______ N. ______ E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar Aneroid 2360' Check before leaving					
Depth to top of bed gave 2370'	0'0"	1'0"	1'0"	13	brown with dk. gray streaks
Elev. of top of bed	1'0"	2'0"	1'0"	14	brown dark moist magnetic soil
Thickness of bed	2'0"	3'0"	1'0"	15	moister, brown, serpentine chunk
Elev. bottom bed	3'0"	4'0"	1'0"	16	very moist, yellowish gray
Depth of hole 7 feet	4'0"	5'0"	1'0"	17	very moist, yellowish gray,
Elev. water table not encountered					quite plastic, Jar no. 38,
Bottomed in peridotite rock					varicolored
***********	5'0"	6'0"	1'0"	18	very moist, yellowish gray
Drill used auger, chopping	6'0"	7'0"	1'0"	19	very moist, gritty, reddish
Number men 3					brown globs and streaks, still
Engr. in charge Libbey					gray bottomed at 7'0" - hit
Mtrl. classfd. by "					peridotite rock.
Sampler "					
Date hole began June 1, 1946					
Date hole finished June 1, 1946					
Shifts actually drilled 1 hour					

Remarks Sample no. 12 is sack of shot collected 30 ft. west of road. 0.19 mi. S. of hole 1. Drilled 2 ft. at site and moved over to south 3 ft. and picked up sampling at 2 feet. Sample no. 20 is rock exposed at hole 2 and sample no. 21 is "shot" collected on surface at hole 2.

WL-24b

Hole No. 2

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite

Project

Hole No. 3 Location Roadcut on west side of Pistol R. road where others have sampled 0.20 mi. south of milepost 14 and 0.45 mi. south of hole no. 1

of sec. T. R. County Curry

Coord. N. E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar 2250' which is 1 ft. above road level					
Depth to top of bed				22	2 ft. channel sample of yellowish
Elev. of top of bed					brown soil, random rock in bank
Thickness of bed					above collar of hole
Elev. bottom bed	0'0"	1'0"	1'0"	23	soft yellowish brown-firm
Depth of hole 7 feet	1'0"	2'0"	1'0"	24	" " " "
Elev. water table	2'0"	3'0"	1'0"	25	" " " "
Bottomed in peridotite rock	3'0"	4'0"	1'0"	26	Jar no. 31 lumps ", still no
***********	4'0"	5'0"	1'0"	27	yellow brown with green spots black
Drill used auger and chopping	5'0"	6'0"	1'0"	28	darker-brown-moist " " spots
Number men 3	6'0"	7'0"	1'0"	29	" " moister" " "
Engr. in charge Libbey					bottomed at 7'0" - peridotite
Mtrl. classfd. by "					rock encountered
Sampler "					
Date hole began June 1, 1946					
Date hole finished					
Shifts actually drilled 1 hour					

Remarks sample no. 22 is overlain by 1 foot of deep brown soil. Mining camp Jct. is 0.55 mi. s. of hole no. 1 and 0.1 mi. s. of hole no. 3. Sample no. 30 is peridotite rock with relatively large magnetite grains. Occurs as float at end of road leading n.w. and down from Jct. with Pistol R. Road and Mining Camp Road; about 1 mile. Sample no. 31 is of

WL-24b

Hole No. 3

(over)

greenstone(?) which occurs as isolated masses in peridotite areas, occurs on road to N.W. in several spots. Impression is that they are roof pendants. The rock is much jointed with fillings in them. Schist containing quartz occurs on hill south of saddle of junction of 3 roads. It is probably a part of the Colebrooke fm. Sample no. 32 is of greenstone N. E. of hole no. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flat Laterite
Project

Hole No. 4 Location on lower flat 0.10 mi. S. 67°E. of house - shop at camp

of sec. T. R. County Curry

Coord. N. E.

Elev. collar 2250' aneroid
Should be corrected

Depth to top of bed

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole

Elev. water table

Bottomed in

****

Drill used

Number men

Engr. in charge Libbey

Mtrl. classfd. by "

Sampler "

Date hole began June 1, 1946

Date hole finished June 1, 1946

Shifts actually drilled

From	To	Thickness of sample	Sample No.	Description
0'0"	1'0"	1'0"	33	chocolate brown soil
1'0"	2'0"	1'0"	34	lighter brown, loose rocks
2'0"	3'0"	1'0"	35	brown rocky with white pieces
3'0"	4'0"	1'0"	36	brown, very rocky (peridotite)
4'0"	5'0"	1'0"	37	brownish gray, clayey
5'0"	6'0"	1'0"	38	brown and gray, clayey
6'0"	7'0"	1'0"	39	moist gray, clayey, Jar 26
7'0"	7'10	10 in.	40	bottomed at 7'10' - hit
				peridotite rock, moist, gray
				and clayey

Remarks

WL-24b

Hole No. 4

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 1 Location Mileage 37.94 (20 ft. E. of Pistol R. Rd. at turnout)
Aneroid reading 2340' (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

Elev. collar ~~2340~~ Aneroid
Should be corrected

Depth to top of bed

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole 11 ft.

Elev. water table not encountered

Bottomed in peridotite rock

****

Drill used auger + chopping

Number men 3

Engr. in charge Libbey

Mtrl. classfd. by

Sampler Libbey

Date hole began May 31, 1946

Date hole finished May 31, 1946

Shifts actually drilled
3PM to 4:30PM

From	To	Thickness of sample	Sample No.	Description
0'0"	1'2"	1'2"	1	deep reddish brown soil,
				weathered peridotite
1'2"	2'0"	10"	2	soft yellow soil
2'0"	3'0"	1'0"	3	begins to be gritty at 2';
				less weathered - yellow, piece
				of small boulder at 2'6"
3'0"	4'0"	1'0"	4	yellow lumpy soil - soft
4'0"	5'0"	1'0"	5	yellow, clayey, very soft,
				greenish black streaks
4'6"	5'0"	0'6"	5A	moisture sample taken
				Jar no. 27
5'0"	6'0"	1'0"	6	yellow to greenish black clay
				soft, moist, clayey, plastic
6'0"	7'0"	1'0"	7	same as no. 6
7'0"	8'0"	1'0"	8	moister more even colored, dark streaks
7'0"	8'6"	1'0"	8A	Jar 30, same as 8

Remarks 38.19 (Mile post 14) (MORE) 38.47 Jct. with road to mining camp

WL-24b

Hole No. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 1-CON'T Location Mileage 37.94 (20' E. of Pistol R. Rd at turnout)

Aneroid Reading 2340 (Should be corrected)

of sec. T. R. County Curry

Coord. N. E.

	From	To	Thickness of sample	Sample No.	Description
Elev. collar					
Depth to top of bed	8'0"	9'0"	1'0"	9	moister, still yellow, brown spots
Elev. of top of bed					clayey, very plastic + wet occasional lumps
Thickness of bed	9'0"	10'0"	1'0"	10	same as 9 with partly weathered
Elev. bottom bed					peridotite pebble, muddy
Depth of hole	10'0"	11'0"	1'0"	11	Jar no. 32, muddy,
Elev. water table					numerous small partially
Bottomed in peridotite rock					weathered peridotite lumps
***********					greenish lumps, still yellow soil
Drill used					Encountered rock at 11ft.
Number men					where hole was bottomed
Engr. in charge					
Mtrl. classfd. by					
Sampler					
Date hole began					
Date hole finished					
Shifts actually drilled					

Remarks

WL-24b

Hole No. 1

Apparently should be 260 feet (45 paces - 5.8 ft. to pace) (Mr. Libbey's notes say 500 ft S45W of pt on road

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

[milepost 14]

Hole No. 2 Location 2600 ft. S.45°W. (to 50°W.) of point on road, 0.19 mi. S. of hole no. 1 + 0.06 mi. N. of (N.25°E. from hole no. 2 to Pyramid Rock) - 200 ft. E. of high point or ridge top of flat where rock crops out - about 30 ft. higher

of sec. ____ T. ____ R. ____ County Curry

Coord. ____ N. ____ E. ____

	From	To	Thickness of sample	Sample No.	Description
Elev. collar Aneroid 2360'					
Depth to top of bed Check before leaving gave 2370'	0'0"	1'0"	1'0"	13	brown with dk gray streaks
Elev. of top of bed	1'0"	2'0"	1'0"	14	brown dark, moist +
Thickness of bed					magnetic soil
Elev. bottom bed	2'0"	3'0"	1'0"	15	moister, brown, serpentine [chunk]
Depth of hole 7 feet	3'0"	4'0"	1'0"	16	very moist yellowish gray
Elev. water table not encountered	4'0"	5'0"	1'0"	17	very moist yellowish gray, quite plastic
Bottomed in peridotite rock					Jar no. 28, varicolored
***********	5'0"	6'0"	1'0"	18	very moist, yellowish gray
Drill used auger, chopping	6'0"	7'0"	1'0"	19	very moist, gritty, reddish
Number men 3					brown globs + streaks, still gray
Engr. in charge Libbey					bottomed at 7'0" -
Mtrl. classfd. by Libbey					hit peridotite rock
Sampler Libbey					
Date hole began June 1, 1946					
Date hole finished June 1, 1946					
Shifts actually drilled 1 hr					

Remarks Sample no. 12 is sack of shot collected 30 ft. west of road, 0.19 mi. S. of hole 1. Drilled 2 ft. at site + moved over to South 3 ft + picked up sampling at 2 feet. Sample no. 20 is rock exposed at hole 2 and sample no. 21 is "shot" collected on surface at hole 2.

WL-24b

Hole No. 2

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Laterite
Project

Hole No. 3 Location Roadcut on west side of Pistol R. road where others have sampled 0.20 mi. South of milepost 14 and 0.45 mi. south of hole no. 1

of sec. T. R. County Curry

Coord. N. E.

Elev. collar 2250' which is 1 ft. above road level

Depth to top of bed

Elev. of top of bed

Thickness of bed

Elev. bottom bed

Depth of hole 7 feet

Elev. water table

Bottomed in peridotite rock

****

Drill used auger & chipping

Number men 3

Engr. in charge Libbey

Mtrl. classfd. by "

Sampler "

Date hole began June 1, 1946

Date hole finished

Shifts actually drilled 1 hour

From	To	Thickness of sample	Sample No.	Description
			22	2 ft. channel sample of
				yellowish brown soil, random rock
				in bank above collar of hole
0'0"	1'0"	1'0"	23	soft yellowish brown - firm
1'0"	2'0"	1'0"	24	" " " - firm
2'0"	3'0"	1'0"	25	" " " "
3'0"	4'0"	1'0"	26	Jar no. 31 ", still no lumps
4'0"	5'0"	1'0"	27	yellow brown with green black spots
5'0"	6'0"	1'0"	28	darker - brown - moist " " "
6'0"	7'0"	1'0"	29	" " moister " "
				bottomed at 7'0" - peridotite
				rock encountered

Remarks Sample no. 22 is overlain by 1 foot of deep brown soil
Mining Camp Jct is 0.55 mi S. of hole no. 1 and 0.1 mi S. of hole no. 3
Sample no. 30 is peridotite rock with relatively large
magnetite grains. Occurs as float at end of road leading
NW and down from Jct with Pistol R. Road & Mining
Camp Road; about 1 mile. OVER

WL-24b Hole No. 3

Sample no. 31 is of greenstone(?) which occurs as isolated masses in peridotite area. Occurs on road to NW in several spots. Impression is that they are root pendants. The rock is much jointed with fillings in them. Schist containing quartz occurs on hill south of saddle at junction of 3 roads. It is probably a part of the Colebrooke fm. Sample no. 32 is of greenstone NE of hole no. 1

STATE DEPARTMENT OF GEOLOGY & MINERAL INDUSTRIES

Red Flats Lakerite
Project

No. 4 Location on lower flat, 0.1 mile S. 67° E. of house-shop at camp

of sec. T. R. County Curry

d. N. E.

. collar 2250' aneroid should be corrected

h to top of bed

. of top of bed

kness of bed

. bottom bed

h of hole

water table

sed in

****

l used

er men

. in charge Libbey

. classfd. by Libbey

ler Libbey

hole began June 1 1946

hole finished June 1, 1946

s actually drilled

From	To	Thickness of sample	Sample No.	Description
0'0"	1'0"	1'0"	33	chocolate brown soil
1'0"	2'0"	1'0"	34	lighter brown, loose rocks
2'0"	3'0"	1'0"	35	brown, rocky with white pieces
3'0"	4'0"	1'0"	36	brown, very rocky (peridotite)
4'0"	5'0"	1'0"	37	brownish gray, clayey
5'0"	6'0"	1'0"	38	brown + gray, clayey
6'0"	7'0"	1'0"	39	moist gray, clayey Jar 26
7'0"	7'10	10 in.	40	bottomed at 7'10" – hit peridotite rock, moist gray + clayey

ks

Hole No. 4

Name: Red Flats Nickel Prospects

Owners: Hanna Mining Company purchased claims in sec. 30, T. 37 S., R. 13 W., about 8 years ago. In April, 1975, Al Wood located claims for Hanna Mining Co. in the N½ sec. 25 and the S½ of the NE¼ Sec. 13, T. 37 S., R. 14 W. The remainder of the area is held by the Red Flats Nickel Corp., Dennis A. Winn, President, P.O. Box 39, Gold Beach, Oregon 97444.

The area is held by a group of contiguous association placer claims.

[illegible] The deposits are in T. 37 S., R. 13 and 14 W. Secs. 18, 19, 29, 30, 31 and 32 in R. 13 W. and 13,24, and 25, T. 37 S., R. 14 W. Access to the area is via the Hunter Creek Road and it is about 23 kilometers from U. S. 101 just south of Gold Beach. The distance to electrical power is about 20 kilometers. Adequate water is located near-by. A central point is located at 42°21'8" N. Lat. and 124°17'54" W. Long.

Elevations range from about 427 meters near Hunter Creek to 858 meters at the highest point in the SW¼ sec. 19.

Climate and vegetation Average annual precipitation is about 208 cm most of which occurs between October and May. Average temperature in summer is about 16°C. and in winter about 7° C. The working season could be 12 months; but is disagreeable during cold rainy periods. The vegetative cover is scrub pine and brush, including knobcone pine, manzanita, huckelberry (both coastal and red) tan oak shrub, azalea, heather, cascara shrub, myrtle shrub, etc. The brush is particularly dense and difficult to traverse on foot in much of the area.

Land use of the surrounding area (other than ultramafic) is chiefly timber production and there is some cattle grazing.

History: Mining claims must have been located sometime in the 1930's. The area was first examined by J. E. Morrison in 1937 at which time he reported a few shallow trenches and a 32-foot shaft. Libbey, Lowry and Mason (1947) explored by hand-auger and reconnaissance mapping in 1946-47. U. S. Bureau of Mines explored the area by preliminary hand-auger sampling and later by Star churn drill (Hundhauser, McWilliams and Banning, 1954). The claim holders have explored by a series of shallow bulldozer trenches.

In 1971 a small bog area near the center of the SE¼ sec. 13 which contains an interesting floral assemblage typical of swampy areas in ultramafic rock was proposed for withdrawal as a special botanical area. As a result, Department geologists Norm Peterson and Len Ramp investigated the area on November 9, 1972 and reported results April 17, 1973 (see attached report).

General geology: The area appears to be characterized by a relatively thin sheet or thrust-plate of ultramafic rock overlying the younger Dothan-Otter Point Formation. The ultramafic rocks are in patches, intermixed with the Colebrooke Schist. Landsliding is common and appears to be an important feature of the deposit in northern, western and southeastern extensions of laterite areas mapped. Lateritic soil and saprolite have developed on the partly serpentinized harzburgite. Areas of intense shearing and complete serpentinization generally contain much less soil cover.

Description of the deposits: Previous descriptions are given by Morrison (1940) Libbey, Lowry, and Mason (1947), Dole, Libbey, and Mason (1948), Hundhausen, McWilliams and Banning (1954), (Appling (1955), Hotz (1964), and Hotz and Ramp (1969).

The areas of reddish-brown to yellowish-brown lateritic soil are generally quite rocky. The rocks appear to be more abundant and the soil cover relatively shallow in areas of steeper slopes while the flats and bench areas have a better accumulation of soil. A few of the upland untransported residual soil areas have a characteristic accumulation of iron "shot" or pellets on the surface. The areas of lateritic soil are plotted on the accompanying map by surface reconnaissance mapping and aerial photo interpretation using color infrared photos taken in 1973 (Project No. 41033) borrowed from the U. S. Forest Service. The overall area of lateritic soil is estimated to be about 300 hectares.

The maximum depth of soil development reported by Hundhausen, et al (1954) is about 15 meters and the average depth about 4 meters (3.85 m). Average depth over the 750 A 300 hectares is estimated to be about 2.5 meters. The average ratio of rock to soil and saprolite is estimated to be about 65:35.

Grade and tonnage estimates: The average grade of soil and saprolite based on a large number of samples is about 0.80 percent Ni, 0.15 percent Co, 1.14 percent Cr_2O_3, and 18 percent Fe. Grade of the mixed soil and rock is calculated to be about 0.41 percent Ni, 0.09 percent Co, and 0.53 percent Cr_2O_3 and 8.94 percent Fe. Estimated gross tonnage of soil and rock using a factor of 1.9 m.t./cu.m. is 14,250,000 tonnes of mixed soil and rock. Net tonnage of soil and saprolite, using a factor of 1.6 m.t./cu.m. is 4,200,000 tonnes. The waste rock in this estimated tonnage would amount to about 10,000,000 tonnes.

17,100,000
7,200,000
243000,000

Using the same grade figures it appears reasonable that one could use a little more optimistic depth and soil:rock ratio figures - up to an average depth of 3 meters and 50 percent rock - and place the resulting tonnage figures in the 25 percent possibility inferred category. These less conservative figures give a calculated potential gross tonnage of soil and rock for the 300 hectare area of 17,100,000 tonnes and 7,200,000 tonnes soil and saprolite.

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Report by: Len Ramp November 18, 1975

Bureau of Mines
Report of Investigations 5072

PRELIMINARY INVESTIGATION OF THE RED FLATS NICKEL DEPOSIT, CURRY COUNTY, OREG.

BY R. J. HUNDHAUSEN, J. R. MCWILLIAMS, AND L. H. BANNING

Dennis A. Winn

United States Department of the Interior — September 1954

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UNITED STATES DEPARTMENT OF THE INTERIOR
Douglas McKay, Secretary
BUREAU OF MINES
J. J. Forbes, Director

September 1954

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PRELIMINARY INVESTIGATION OF THE RED FLATS NICKEL DEPOSIT, CURRY COUNTY, OREG.

by

R. J. Hundhausen,[1] J. R. McWilliams,[1] and L. H. Banning[2]

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2/ Chief, Ferrous Metals Branch, Metallurgical Division, Region II, Bureau of Mines.

Report of Investigations 5072

INTRODUCTION AND SUMMARY

The continued worldwide shortage and strategic value of nickel place great importance upon the continued development and exploitation of new sources of nickel in the United States. The deposits of nickel-bearing minerals in Oregon and Washington have attracted considerable attention, particularly since the Hanna Development Co. has engaged in development work aimed at producing ferronickel from the Nickel Mountain deposit, near Riddle, Douglas County, Oreg.

The Red Flats nickel deposit in Curry County, Oreg., was first examined by the Bureau of Mines in June 1945, but it was not until 1952 that the deposit was drilled and sampled. In 1953, a 15-ton sample of Red Flats material was procured and subjected to continuous smelting tests in the Bureau's Northwest Electrodevelopment Laboratory, Albany, Oreg.

The sample of Red Flats nickel ore as received contained 31 percent moisture. It was dried to 18-percent moisture and stage-crushed to minus-1/2-inch for the smelting test. The ore charged to the furnace contained 0.74 percent nickel and 28.2 percent iron. The furnace charge was in the ratio of 100 pounds of ore to 17 pounds of wood chips. No fluxing constituents were added. Selective reduction and an arc smelting technique were used throughout the test. Overall recovery figures indicated that 82.2 percent of the nickel and 12.8 percent of the iron charged to the furnace were recovered in ferronickel products analyzing 9.29 to 14.0 percent nickel. The average power and graphite consumption was 1,073 kw.-hr. and 10.52 pounds of graphite, respectively, per ton of dry ore smelted.

Neither the field investigations nor the smelting tests are considered complete; nevertheless, a fairly large area of nickeliferous laterite and nickeliferous serpentine is indicated by the Bureau's drilling program; the preliminary smelting tests show that it is technically feasible to recover a low-carbon ferronickel product from the Red Flat ore.

Mining the deposit presents no special problems. Open-pit methods after stripping overburden would be satisfactory. Transportation of ore to railhead, seaport, or the nickel smelter at Riddle will be expensive and, considering the low grade of the ore, will be perhaps the largest handicap to utilization.

PREVIOUS WORK AND ACKNOWLEDGMENTS

The Red Flats area has been examined by the Oregon State Department of Geology and Mineral Industries several times, the first examination being in 1937. The deposit came to the attention of the Bureau of Mines in June 1945, when Bureau engineers[3] made a preliminary examination and drilled 16 hand-auger holes.

3/ H. G. Iverson and R. J. Hundhausen, mining engineers, Bureau of Mines, Mining Division.

In 1946 the Oregon State Department of Geology and Mineral Industries initiated a project on the laterite soils of southwestern Oregon and began[4] the work at the Red Flats deposit. In 1947 the department did some additional sampling.[5]

Special acknowledgment is due F. W. Libbey, director of the Oregon State Department of Geology and Mineral Industries, for his suggestions and criticism in preparing this report.

Funds for the mining investigations were provided by the California-Oregon Power Co., Medford, Oreg., in the interest of mineral development of southwestern Oregon. The metallurgical investigation was financed by funds from the project entitled, "Research on usability of Fe-Ni-Cr ores in Oregon and Washington."

This report describes the field investigations conducted by J. R. McWilliams[6] under the supervision of D. W. Butner[6] and the metallurgical investigations under the immediate supervision of Wallace E. Anable[7] and directed by L. H. Banning, chief, Ferrous Metals Branch.

LOCATION AND ACCESS

The deposit is in sec. 19 and 30, T. 37 S., R. 13 W., 8 miles airline, or 17 miles by road, east of Gold Beach, Curry County, Oreg., in the extreme southwest corner of the State. The Oregon coast highway (U.S. 101) follows along the narrow coastal plain connecting the small towns of Brookings, Gold Beach, and Port Orford. The rugged upland interior is accessible by a few Forest Service roads and is sparsely populated. The Red Flats deposit is on the narrow, unsurfaced Forest Service road leading to the Snow Camp Mountain lookout. This road leaves U. S. Highway 101 at Hunters Creek, 3 miles south of Gold Beach (fig. 1).

The nearest railroad connection is at Coquille, Oreg., which is 77 miles north of Gold Beach and 94 miles by road from the deposit. Transportation is a serious handicap to development of mineral resources in this relatively inaccessible region of Oregon.

HISTORY AND PRODUCTION

The Red Flats deposit was located as a gold-mercury placer prospect in the early thirties. Small mills and retorts have been set up on the property at different times. A few shallow hand trenches and a 32-foot shaft comprised

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4/ Libbey, F. W., Lowry, W. O., and Mason, R. S., Nickel-Bearing Laterite, Red Flat, Curry County, Oreg. The Ore Bin, vol. 9, No. 3, March 1947, pp. 19-27.

5/ Dole, Hollis, Libbey, F. W., and Mason, R. S., Nickel-Bearing Laterite Areas of Southwestern Oregon: The Ore Bin, vol. 10, No. 5, May 1948, pp. 33-38.

6/ Mining engineers, Bureau of Mines, Mining Division.

7/ Metallurgist, Bureau of Mines, Metallurgical Division, Region II.

the early development work. In 1945 and 1946, 4 or 5 shallow bulldozer trenches were excavated, and 3 or 4 small camp building were constructed by the operators. The property has been relatively dormant since 1947.

None of the operations in the past have proved successful, and most of the equipment has been removed. There is no record of any production from the property.

PROPERTY AND OWNERSHIP

The deposit is covered by several groups of claims, the boundaries of which are not accurately known. The largest area claimed consists of 9 unpatented placer claims, covering 1,600 acres, owned by the Red Flats Association, of which Mrs. ~~Mary Smedburg,~~ Gold Beach, and J. A. Walsh, Coos Bay, are the principal representatives.

The Red Ridge Mining Co., represented by Harry Hedderley and Associates of Gold Beach, owns a group of unpatented placer claims adjoining the Red Flats Association property on the north. In 1943, the Pistol River Mercury Mining Co., James Gensler, trustee, Portland, Oreg., controlled a large group of placer claims known as the Glade Creek Placer Association property. This property adjoins that of the Red Flats Association on the east.

A number of fringe claims are owned by various individuals and groups.

PHYSICAL FEATURES AND CLIMATE

The Red Flats deposit is on a relatively flat topped ridge between Hunters Creek on the west and Pistol River on the east. The lower slopes of the ridge are deeply dissected, rugged, and steep. The summit of the ridge is terraced or benched, but undissected. The altitude of the deposit ranges between 2,150 and 2,500 feet above sea level.

Large areas of the deposit are relatively barren of timber and covered only by dense brush. Scattered patches of knob cone pine, stunted oaks, and manzanita are elsewhere on the deposit but have little value for mining purposes. West of the deposit, timber of excellent quality is found in large quantity.

Typical Oregon coastal climate prevails with heavy rainfall, fog, and storms common during the winter and spring months. Snow and freezing temperatures are exceptional. Annual precipitation averages about 50 inches. The summers are dry and mostly clear, and there is little temperature variation, the yearly mean average temperature being around 50° F.

Water is available from Pistol River or Hunters Creek at all times. The higher slopes of the ridge are comparatively dry. One permanent spring near the summit of the ridge supplies enough water for domestic and limited industrial use.

DESCRIPTION OF DEPOSIT

General

The Red Flats nickel deposit contains areas of nickel-rich brick-red soil, termed nickeliferous laterite, and an underlying zone of weathered nickeliferous serpentine. There are large differences in chemical composition between the two types of nickeliferous materials.

Nickeliferous Laterite

Geologic Setting

The general geology of this region is described by Butler and Mitchell 8/

The nickeliferous laterite rests on a weathered bedrock of ultrabasic rocks, principally peridotite and serpentinized peridotite. These rocks are part of the Josephine peridotite intrusives common in southwestern Oregon and northern California.9/ The nickeliferous laterite is confined to the surface of the peridotite and has been derived from it under certain conditions of weathering, particularly where there are alternating cycles of hot dry climate followed by a prolonged rainy season. Where the peridotite contacts Colebrooke schists (pre-Jurassic) and the Dothan formation (Jurassic) immediately south of the Red Flats area, the laterite is absent.

The comparatively low iron content and the high magnesium and silicon contents of the laterite in the Red Flats area indicate a relatively youthful stage of weathering. If more mature laterites, high in iron content, were ever developed in this area, they have since been eroded, and there is nothing to indicate their former presence. The iron content of the laterite ranges in grade from 25 to 45 percent. Residual boulders of undecomposed peridotite are scattered throughout the laterite. The laterite has accumulated along the axis of the ridge and in small basins on the east slopes sheltered or rimmed by resistant dikes or ledges. The west slope of the ridge is covered with only a thin veneer of red soil (laterite).

Nickeliferous Laterite Areas

The nickel-rich laterite areas have not been completely delimited by the Bureau of Mines drilling program. Apparently more of this laterite is present north of the area explored. The nickeliferous laterite areas are outlined in figure 2. A cutoff grade of 0.90 percent combined nickel and cobalt was used to establish the boundaries of higher grade laterite areas. The

8/ Butler, G. M., and Mitchell, G. J., Preliminary Survey of the Geology and Mineral Resources of Curry County, Oreg.: Oregon Bureau of Mines and Geology, Mineral Resources of Oregon, vol. 2, No. 2 October 1916, 136 pp.

9/ Wells, F. G., Hotz, P. E. and Cater, F. W., Jr., Preliminary Description of the Geology of the Kerby Quadrangle, Oregon: Oregon State Department of Geology and Mineral Industries Bull. 40, 1949, 23 pp.

depth of nickel-rich laterite and the grade are shown for each hole. None of the areas contain overburden except hole 17, in which 5 feet of relatively barren laterite is on top of the nickel-rich laterite.

The shape and form of the nickel-rich laterite areas are variable; the nickel content of the laterite is variable, and some of the material is a mixture of residual laterite and transported laterite.

The nickel content of the laterite cannot be estimated by visual examination but must be determined by analytical methods. The nickel content does not necessarily depend on the degree of weathering or the depth of laterite soil. A few surface indications have been recognized that roughly aid in prospecting for nickel-rich laterites, but these are not infallible guides. Surface accumulations of rounded, shot-size pellets of iron oxides in the soil are fair indicators of a relatively higher nickel content in the laterite. Furthermore, a loosely consolidated, porous, soft, well-drained, residual-type laterite generally contains more nickel than a clayey, dense, compact, transported-type laterite. Residual laterite usually has a higher specific gravity than a clayey type or transported laterite.

Chemical and Mineralogical Composition of Laterite

No discrete nickel minerals were observed in the laterite. The nickel is believed to be in a finely divided form as hydrated nickel oxides; some is chemically combined in the limonite mineral structure; a little may be with magnetite and chromite. It is not possible to improve the grade of nickeliferous laterite by handsorting nor has beneficiation by ore dressing been successful to date.

The chemical composition of nickeliferous laterites is surprisingly uniform throughout the world. These laterites are only formed on or derived from serpentinized ultrabasic rocks, which apparently are relatively uniform in composition. The nickeliferous laterites are quite different in composition from laterites derived from the more highly differentiated silicic rocks.

Serpentine rocks generally lack vegetation. A serpentine rock must undergo a certain amount of chemical decomposition before mechanical disintegration and laterite formation. The ultramafic silicate minerals, such as olivine, have iron, nickel, chromium, aluminum, calcium, and magnesium tied up in their chemical structures. These minerals lack chemical stability under surface weathering, oxidizing conditions, but they do not disintegrate immediately. They hydrolyze and become partly hydrated and recrystallize, forming more stable hydrous silicates. This is a gradual rock-softening process, with very little change in net composition. As this reaction proceeds, the soluble bases and silicic acid are more easily removed by solution in ground water, and more complete chemical decomposition follows. Colloidal-size particles then may be removed rapidly by mechanical means, together with the chemical solution of the soluble bases. Under favorable topographic and climate conditions, the insoluble bases and heavy minerals accumulate to form residual laterite deposits.

degree of residual enrichment is obtained, the nickel no longer accumulates but instead is leached out of the soil, judging from the fact that many of the enormous nickel laterites of the world have uniform nickel contents averaging close to 0.6 or 0.7 percent nickel. Assuming that most serpentines average close to 0.10 percent nickel over large areas, the ratio of concentration of nickel in the soil due to weathering is generally about 6 or 7 to 1. Some relatively small areas of serpentine average 0.3 to 0.5 percent nickel. When these areas are laterized, the nickel content in the resulting residual laterite may accumulate to an average maximum of 1.8 to 3.5 percent nickel, calculated at the same ratio of concentration as before. These richer areas of laterite constitute the minable deposits in Cuba. Most residual laterites show variations in nickel content that reflect to a marked degree the primary differences in the nickel content of the serpentinized rock that has been weathered. Transported laterite of course would not exhibit this relationship with the underlying bedrock.

Nickeliferous Serpentines

The nickel-rich serpentinized bedrock in the Red Flats area, as determined by assays of the Bureau of Mines drill-hole samples, is outlined on figure 2. A cutoff grade of 1 percent combined nickel and cobalt was used to delimit the boundaries. The depth of overburden on this nickeliferous serpentine is shown for each hole. Overburden may consist of barren laterite, nickel-rich laterite, or relatively barren rock.

Lack of development of this bedrock deposit precludes accurate description. The nickeliferous area may be more irregular than is indicated. The general trend of the nickeliferous zone is in a northerly direction, coincident with the axis of the ridge. This zone has an explored length of 3,900 feet, as determined by the drilling; the zone may extend farther to the north beyond the area explored. The extension of the zone to the south is limited. The average width is 590 feet and the average thickness is 12.5 feet.

Garnierite (hydrous nickel silicate) is exposed in the serpentinized peridotite in the trench by drill hole 15 near the south end of the zone. This trench is a few hundred feet north of the permanent springs that supply the Red Gold mining camp. The moist soil in the vicinity of the springs supports a luxuriant growth of Darlingtonia (pitcher, cobra, or fly-catcher plants). Garnierite was found at Woodcock Mountain in a similar environment just above a side hill springs area supporting a conspicuous growth of Darlintonia. These plants may be indicators of nickel-rich zones in the serpentinized ultrabasics of southern Oregon.

FIELD WORK BY THE BUREAU OF MINES

Drilling

The Red Flats area was drilled on a grid system using a Star churn drill mounted on a 6-wheel-drive truck. Twenty-two holes, 6 inches in diameter and ranging from 20 to 117 feet and averaging 35 feet deep, were drilled in 8 lines 500 feet apart. The holes were drilled in order of their accessibility, and this accounts for the somewhat erratic numbering system. The cutoff point in drilling was generally determined by the increased hardness of the rock, discernible in drilling.

The area was surveyed with a Brunton compass and tape. Elevations were obtained by rod and hand-level method.

Sampling

The drill holes were sampled in 5-foot sections. A thin slurry of sludge was maintained by water control. The sludge volume per section varied from 4 gallons near the surface to 14 to 20 gallons below 10 feet. Each sample was split at the drill site. Samples were dried over an open fireplace and split again, one split being submitted for analyses and the other split again. One of the second splits was panned in the field, and the other was saved to combine with other rejects from the same hole to provide a composite sample for analyses. Composite samples from four holes were sun-dried and assayed for mercury.

At the beginning, each 5-foot sample was assayed for nickel and gold; early returns showed no gold detected, consequently gold assays were discontinued. A number of the nickel-rich samples were assayed for Fe, Mg, Cr_2O_3, and Co. The complete assays of the drill holes are shown in the appendix.

To check the mercury content, a composite sample was treated by table concentration to produce a concentrate and a tailing. Assays of concentrates showed less than 1 pound of mercury per ton. Assuming a recovery of 100 percent, the mercury content of the original material would be less than 0.00002 pounds per ton.

METALLURGICAL RESERACH BY BUREAU OF MINES

General

The Red Flats deposit contains both nickeliferous laterite and nickeliferous serpentine. The 15-ton sample used in the smelting test was apparently a mixture of these two types of material as the SiO_2 content was higher than most laterites and the MgO content was lower than most serpentines.

Nickel is recovered from a mixture of Cuban laterites and serpentines in a plant at Nicaro, Cuba, by a roasting and ammonia-leaching process.[10] Only limited amounts of serpentine are used in this process because plant operating difficulties arise when excessive amounts are used.

Electric smelting research at Albany, Oreg., has indicated that nickel can be recovered from either laterites or serpentines as a low-carbon ferronickel product. This smelting research has been described in a previous publication.[11]

10/ Caron, M. H., Ammonia Leaching of Nickel and Cobalt Ores: Trans. Am. Inst. Min. and Met. Eng., vol. 188, January 1950, pp. 67-90.

11/ Cremer, Herbert, Continuous Electric Smelting of Low-Grade Nickel Ores: Bureau of Mines Rept. of Investigations 5021, 1954, 36 pp.

The object of the smelting test on Red Flats material was to determine whether or not the selective reduction technique developed by the Bureau for the production of low-carbon ferronickel from Riddle and Cle Elum ores could be successfully applied to the Red Flats ore.

Equipment and Facilities for Smelting Research

The Northwest Electrodevelopment Laboratory, Albany, Oreg., has a laboratory completely equipped for electric smelting research. In addition to two steelmaking furnaces and other necessary auxiliary equipment, the laboratory has two 3-phase electric smelting furnaces of Bureau design. The larger smelting furnace uses 6- or 8-inch graphite electrodes at power inputs of between 400 and 1,000 kw. The smaller furnace uses 3- or 4-inch graphite electrodes at power inputs ranging from 100 to 300 kw.

Materials Handling

When the sample of Red Flats nickel ore was received at Albany, the four truckloads were dumped into separate piles on a large concrete storage pad adjacent to the smelter building. A Ford tractor equipped with a Dearborn industrial-type loader was used to transport the ore from the storage pad to the feed end of the drier. The rock was hand-sorted from the earthy material, weighted, sampled, and analyzed. The earthy material was dried in a 23-foot long by 18-inch-diameter rotary-type propane gas-fired drier. Figure 3 shows the drier in operation. The dried material was discharged into a raw materials bin, then fed through an 8- by 15-inch jaw crusher, and down a chute to a bucket elevator, which was used to elevate the material onto a vibrating screen with 1/2-inch square openings. The screen undersize was sampled by an automatic sampler and discharged onto an endless conveyor belt for distribution into one of six 25-ton, steel storage bins. The screen oversize was funneled into a model CF-3-26 Pennsylvania impactor. The discharge from the impactor was combined with the jaw-crusher discharge and returned to the bucket elevator for recycling over the screen. Each load was dried, crushed, and sampled separately, drawn from the receiving bin, and bedded in a crib in successive layers in the smelter building. The furnace charge was made up of bedded ore and wood chips. The ore and reductant were weighed and mixed in a 6-cubic foot concrete mixer. This operation is illustrated in figure 4. The mixed charge was loaded into a charge bucket, hoisted to the charge deck, and stored adjacent to the furnace. The charge was then hand-shoveled into the furnace at regular intervals.

Description of Smelting Furnace

The continuous smelting test on Red Flats nickel ore was conducted in the small furnace, designated "ESA". This furnace is a round, open-top, pit-type, 3-phase unit. The electrode clamps are cable-suspended with special provisions for readily changing the electrode spacing. The graphite electrodes are placed at the corners of an equilateral triangle. The furnace is backed by a 1,000-kv.-a. Westinghouse transformer, the secondary busses of which may be connected in parallel of series. The transformer has six voltage taps, the series-connected voltage being approximately twice the parallel-connected voltage.

The open circuit phase-to-phase voltages range from 38 to 106 volts with parallel connection and from 80 to 220 volts with series connection.

The steel shell of the ESA furnace has a 50-inch inside diameter and is divided into two sections. The bottom section is 31 inches high and is provided with two tap holes at different levels; the lower tap hole is designed to drain the furnace. The top section is 24 inches high and, with its refractory lining, is removable as a unit. The lower section is lined with magnesite brick to form a 32-inch-diameter smelting crucible. The hearth is rammed magnesite.

Rock conteins same percent Ni%? Co

Description of Ore and Reductant

Four truckloads of Red Flats nickel ore were taken from an area adjacent to three drill holes selected by the Mining Division to be representative of the deposit. The identity of each of these samples was maintained until the ore was dried, sampled, crushed, and bedded in a crib in the smelter building. The ore as received consisted of a small amount of rock mixed with weathered earthy material. The rock was hand-sorted from the first load, in an attempt to remove gangue material and upgrade the ore. A composite sample of this rock assayed only 0.39 percent Ni. Therefore, the rock was sorted out of the three subsequent loads and was not used in this smelting test. However, a composite sample of all rock removed from the 4 loads assayed 0.77 percent nickel; this is equivalent to the nickel content of the earthy material used in the smelting tests. The weight and nickel content of the rock removed from the four truck loads are shown in table 2. In actual practice it would not be necessary to remove the rock from the earthy material because nickel may be removed from either one or both types of ore in the smelting furnace.

Analyses of Soil and Rock

The drill holes from which each load of ore originated, the dry weights, and Ni analyses of the earthy material and rock are shown in table 2.

TABLE 2. - Weights and analyses of Red Flats ore as received, dry basis

Load	Drill hole	Earthy material			Rock		
			Percent			Percent	
		Weight, pound	Ni	Fe	Weight, pound	Ni	Fe
1	15 and 16	5,861	0.68	42.0	533.3	0.39	14.6
2	15	8,071	.86	31.3	1,022.4	.67	10.0
3	16	8,714	.78	17.4	770.3	.72	9.3
4	23	6,415	.84	25.1	1,416.0	.98	13.1

Table 3 gives complete analyses on Red Flats nickel-ore samples. The first earthy material sample is a weighted composite of the four truck load samples taken by the automatic sampler. The second sample was taken by saving a small scoop of earthy material, in a closed container, as each 300 pounds of ore was weighed into the furnace charge. The analyses of the composite Red Flats rock sample closely resemble that of typical serpentines. Complete chemical analyses on Riddle lot F ore are included for comparison. Spectrographic analyses on the composite sample of Red Flats rock, Red Flats earthy material, and Riddle lot F nickel ores are shown in table 4.

TABLE 3. - Chemical analyses of composite samples, percent

Ore	Fe	Ni	Co	Al_2O_3	Cr_2O_3	CaO	MgO	SiO_2	S	P	L.O.I.
Red Flats1/.....	29.8	0.70	0.06	6.2	2.53	3.3	8.6	24.2	0.03	0.18	11.4
Red Flats2/.....	28.2	.74	.07	6.4	2.44	3.2	4.6	28.2	-	-	10.2
Red Flats, rock.	12.7	.77	.06	2.0	.90	3.2	26.9	37.2	.01	.08	10.6
Riddle, lot F...	12.0	1.4	-	3.5	1.05	1.0	19.2	48.1	.01	.04	7.8

1/ Earthy material, composite of 4 truckloads.
2/ Earthy material, head sample for smelting tests.

TABLE 4. - Spectrographic analyses of composite samples

Ore	Fe	Ni	Al	Cr	Ca	Mg	Si	Mn	Co	Ti	K	V	Na
Red Flats, rock	A	D-	C	D	E	A	A	E-	D	E	D	E	-
Red Flats, soil	A	D-	C	C	E	C	B	D	D-	E	D	E	E
Riddle, lot F..	A	C	C	C	D	A	A	D	E	E	-	F	-

Legend: A- over 10 percent; B- 5 to 10 percent; C- 1 to 5 percent; D- 0.1 to 1 percent; E- 0.01 to 0.1 percent; F- 0.001 to 0.01 percent

Petrographic Analysis

Petrographic examinations of the test sample revealed that the rocky material consists of nickeliferous serpentine, with some highly altered fragments of peridotite and less amphibolite. Associated with the serpentine are relatively small amounts of chrysotile, chromite, magnetite, and a trace of pyrolusite. The altered fragments of peridotite and amphibolite contain serpentine, olivine, ortho- and clinopyroxene, amphibole, chlorite, feldspar, limonite, minor amounts of quartz, chromite, and magnetite. The serpentine and rock fragments are coated with a limonitic clayey soil.

The earthy material is similar mineralogically to the rocky material, except that it contains much more of the limonitic soil.

No discrete nickel mineral was observed in either of the samples. Spectroscopic examinations of selected concentrates showed minor amounts of nickel in the serpentine, peridotite, magnetite, and limonitic soil.

Reductant

Previous experience in smelting Riddle and Cle Elum nickel ores indicated that these ores could be smelted selectively best by using hogged fuel as the reductant. The hogged fuel not only provided the carbon necessary for reduction but also maintained a porous, nonconducting, heat-insulating charge over the molten bath. The porous nature of the charge allowed gases generated in the smelting reaction to escape freely, thus preventing violent gas blows and loss of dust and metal vapors.

Hogged fuel is a mixture of wood chips, sawdust, and splinters resulting from feeding ends of logs, slabs, and other waste products of the lumber industry through a machine called a hog. Hogged fuel is generally used for industrial and home heating. The wood chips used in this test contained no sawdust and were obtained from a local plywood plant. They were produced by feeding waste plywood veneer through a machine called a chipper. They were uniform in size, being about 1/8 inch thick and 5/8 inch square. The wood chip charge, used in smelting Red Flats nickel ore, promoted a uniform smelting operation.

Smelting Procedure

The furnace charge proportions were 100 pounds of ore to 17 pounds of wood chips. No fluxing constituents were used. The wood chips were calculated to furnish 2 pounds of fixed carbon. This was 48.5 percent of the stoichiometric carbon requirement for reduction of all nickel, iron, and chromium oxides in the ore and was equivalent to 12.8 times the amount necessary to reduce the nickel; it reduced enough iron to yield a ferronickel alloy containing 10 to 15 percent nickel.

The test was begun by covering the magnesite hearth with 200 pounds of ore; then 4.7 pounds of minus-1/2-inch petroleum coke was placed under the electrodes. The electrodes were lowered onto the coke, and a steady arc was established. The ore under the coke gradually melted and formed a molten pool on the furnace hearth. Charge was then fed around the periphery of the furnace until a dry top was established over the entire molten pool. The charge was banked against the wall of the furnace to such an extent that it rolled down a steep incline to fan out between the electrodes. A shallow dry top was maintained on the molten pool at all times.

An arc-resistant, dry-top smelting technique was used throughout the test; that is, the heat for the smelting reaction was furnished by arcing on the molten bath and the arcs were covered with charge at all times. The phase-to-phase voltage was maintained at 200, and the current averaged approximately 650 amperes per phase. Occasionally some wet charge caved into the smelting zone, cooling the slag and decreasing its electrical conductivity. The electrodes then lowered and dipped into the slag causing boiling and considerable turbulence in the furnace. These occurrences were minimized by maintaining a shallow but uniform dry-top.

Slag was tapped from the upper tap hole at intervals of about 1-1/2 hours. The ferronickel was tapped from the drain tap hole at the end of each shift. The tap holes were usually opened with an iron rod. Occasionally an oxygen lance had to be used when solidified slag and metal blocked the tap holes. The slag was tapped into a conical cast-iron mold of about 400 pounds capacity (fig. 5). An iron hook was frozen into the slag to facilitate weighing and handling. The slag was sampled by dipping a cast-iron spoon into the molten slag and pouring its contents onto a clean graphite plate. Ferronickel and a small amount of slag were tapped from the drain tap hole into a 12-inch-diameter ladle, the supernatant slag was decanted into a conical, cast-iron mold, and the alloy was poured into cast-iron pig molds. Each alloy pig weighed about 15 pounds. The alloy pigs were sampled by drilling.

Discussion of Test Results

The test was divided into four periods. The complete results are presented in table 5. The results of a continuous pilot-plant test on Riddle ore are included for comparison. Riddle test results were described in an earlier publication[12] and are not discussed here.

The smelting technique used for selective reduction of nickel from the Red Flats ore was similar to that used in smelting Riddle ore, except that wood chips were used as the reductant. A more uniform furnace operation was evident with wood chips.

The results show that the nickel content of the ferronickel, as well as the nickel content of the slag, increased progressively during the four periods of the test. The highest nickel-content alloy, 14 percent, was produced during the fourth period. The slag with the lowest nickel content, 0.07 percent, was produced during the first period. These results indicate, as have previous test results, that nickel recovery decreases as the nickel content of the ferronickel product increases. Additional research to definitely establish the relationship between the nickel content of the metal product and the slag when these low-grade nickel ores are smelted would be worthwhile.

In the tabulated results, nickel recovered in the various periods does not include the nickel that gradually accumulated in the furnace or the molten ferronickel that slowly penetrated the porous, rammed magnesite hearth. However, the data indicate that these conditions were coming to equilibrium. In the fourth period, 85.3 percent of the nickel charged to the furnace was accounted for in the slag and metal products. At the conclusion of the test, a large salamander was removed from the furnace. It contained an appreciable amount of nickel, and 94.5 percent of the nickel charged to the furnace was then accounted for.

In previous smelting tests there had been some difficulty in accounting for the nickel charged to the furnace. Therefore, in this Red Flats test the accuracy of the spoon-sampling procedure was checked by comparison with several composite slag cone samples. A spoon sample analyzed 0.13 percent nickel. For the same period, a composite slag cone sample analyzed 0.14 percent nickel. To determine whether or not nickel was concentrating in the lower portion of the slag mold, the lower 6-inch zone of a number of slag cones was collected, crushed, sampled, and analyzed. This slag sample analyzed 0.21 percent nickel and 32 percent iron. Examination of this slag under a binocular microscope revealed numerous ferronickel prills, ranging from the smallest visible particles up to 1/8 inch in diameter. These prills were concentrated near the outer surface of the slag cones. It is reasonable to assume that the nickel contained in the prills was not necessarily accounted for in the slag spoon samples. Nevertheless, 94.5 percent of the nickel charged to the furnace was accounted for in the slag and metal products.

A refractory magnesite-brick lining was used in the furnace for this test. The 1 inch of insulation material normally placed between the brick refractory and the steel furnace shell was omitted to permit more effective cooling of the refractory and thus minimize refractory erosion.

12/ See work cited in footnote 11.

TABLE 5. - Comparison of operating data for smelting nickel ore from Red Flats and Riddle deposits

	For Red Flats ore					For Riddle ore										
Periods	1	2	3	4	Whole run	1	2	3	4	5	6	7	8	9	10	Whole run
Lb. charged per 100 lb. ore																
Wood chips 2/	17	17	17	17												
Sawdust 1/								12	8	6	6	10	8	18		
Coos Bay char						5.5	1.67									
Coos Bay coal															2.5	
Alsifer											0.33	0.33	0.33			
Materials charged, lb.																
Riddle ore						38,100	21,600	11,700	11,100	13,800	10,800	11,100	11,400	9,000	10,695	149,295
Red Flat ore, dry	4,830	9,240	9,830	5,764	29,664											
Sawdust								1,404	888	810	648	1,110	912	1,620		7,392
Wood chips	950	1,950	2,000	1,100	6,000											
Coos Bay char						2,057	350									2,407
Coos Bay coal															253	253
Alsifer											36	37	38			111
Ratio of stoichiometric C 3/	13.7	12.4	12.0	13.2		12.3	3.0	3.2	1.8	1.0	1.6	2.9	2.3	4.9	2.6	
Duration of test, hr.	12.3	23.6	24.3	11.7	71.9	72.1	47.6	27.0	22.2	27.7	21.3	22.8	23.7	23.7	23.7	311.8
Feeding rate, lb. ore/hr.	394	392	405	503	409	528.4	453.8	433.3	500.0	498.2	507.0	486.8	481.0	379.7	451.3	478.8
Power conditions																
Voltage, phase-to-phase	200	200	200	200		205	205	205	205	205	205	205	205	205	205	
Current per phase, amp.	615	615	624	758	653	652	546	567	612	545	592	586	574	563	571	
Power input, kw.	211	213	216	260	225	231	194	201	217	193	210	208	203	200	203	
Power factor	99	100	100	99		99	99	99	99	99	99	99	99	99	99	
Power consumption, kw.-hr.																
Per period	2,590	5,020	5,250	3,058	15,918	16,680	9,220	5,430	4,820	5,350	4,470	4,740	4,820	4,730	4,800	65,060
Per ton of dry ore	1,072.4	1,086	1,068	1,058	1,072	876	854	928	869	775	828	854	846	1,051	898	872
Per lb. of metal	12.3	21.9	50.0	15.6	13.9											9.0
Per lb. of Ni in metal	131.3	203.0	378.8	111.4	88.1											46.7
Electrode consumed, lb.																
Per period					156 4/	363.9	126.0	69.5	55.5	63.3	53.5	50.3	55.3	77.0	74.3	988.6
Per ton of dry ore					10.5	19.1	11.7	11.9	10.0	9.2	9.9	9.1	9.7	17.6	13.9	13.2
Per lb. of metal					.14											.14
Per lb. of Ni in metal					.87											.71
Products, lb.																
Metal	210	229	105	195.5	1,444.5	3,711	442	681	55	60	25	412	128	1315	371	7,199
Slag	3,683	6,834	6,898	4,109	21,524	24,059	16,257	8,470	8,595	10,679	8,555	8,336	9,001	6,032	8,336	108,322
Metal analyses, percent																
Fe	86.80	86.80	85.50	84.50		74.21	65.80	75.60	56.86	48.49	46.23	54.00	54.00	76.63	73.10	
Ni	9.39	10.8	13.2	14.0		8.90	25.20	23.0	40.8	48.20	50.00	42.50	45.20	30.29	27.30	
Cr	1.41	.19	.07	.04		2.99	.96	.25	.13	.08	.03	.05	.03	.18	.08	
C	.06	.06	.17	.10		1.74	.65	.17	.06	.07	.11	.04	.02	.06	.07	
Si	.93	.09	.02	.02		8.50	7.05	.05	.02	.03	.06	.10	.01	.03	.02	
S	.14	.20	.01	.01		.04	.02	.10	.12	.19	.32	.29	.13	.14	.23	
P	.23	.27	.20	.19		.11	.21	.09	.28	.19	.31	.34	.28	.31	.18	
Co	.26	.24	.29	.22												
Slag analyses, percent																
Fe	27.3	29.0	33.0	37.3		3.47	11.39	12.07	13.95	15.06	14.97	13.71	14.00	9.36	10.03	
Ni	.07	.08	.14	.22		.05	.31	.09	.26	.56	.46	.18	.26	.08	.14	
Co	<.02	<.02	<.02	<.02												
Nickel accounted for, percent																
In the metal	55.17	36.16	22.08	64.15	82.23 5/	61.91	36.79	95.56	14.57	13.97	7.78	105.23	33.89	185.01	63.08	64.77
In the slag	7.22	8.00	13.28	21.19	12.28	2.21	16.67	4.66	14.38	28.89	24.29	9.01	13.68	3.71	7.27	11.45
Total in slag and metal	62.39	44.16	35.36	85.34	94.51	64.12	53.46	100.22	28.95	42.86	32.07	114.24	47.57	188.72	70.35	76.22

1/Average fixed carbon 10.1%.
2/Average fixed carbon 11.3.
3/Ratio of stoichiometric C for reduction of the Ni in the ore.
4/Electrode consumption data was determined for the whole run.
5/Includes 705 lb. of furnace and ladle cleanout.

Refractory erosion is a serious problem when a highly siliceous charge is smelted in a basic lining. Only the earthy material, containing 7.75 percent CaO plus MgO, was smelted in this test. The CaO plus MgO content of the rocky portion was 30.1 percent. If nickel is to be recovered economically from this deposit, the rocky portion, of sufficient Ni content, might be blended with the earthy material to produce a neutral slag and minimixe the fluxing effect of the molten slag on the basic furnace lining.

CONCLUSIONS

Further drilling and exploration should be planned to delimit areas of nickel-rich laterite and serpentine indicated by the investigations reported herein.

The results of the 72-hour continuous test indicated that the selective smelting procedure developed by the Bureau for recovering nickel from siliceous nickel ores is applicable to the ore from the low-grade Red Flats deposit.

Highest nickel recoveries were obtained when producing a ferronickel containing 9 to 10 percent nickel. Previous Bureau tests indicate that this grade of ferronickel can be upgraded to any desired nickel content by blowing with air, or a mixture of air and oxygen, in a converter.

Eighty-two percent of the nickel charged to the furnace was recovered in the 1,444.5 pounds of ferronickel produced during the test. Only 5.49 percent of the nickel charged to the furnace was unaccounted for. These results are considered satisfactory for preliminary testing. Indications are that even better recovery could be anticipated if the nickel content of the ferronickel product is kept below 10 percent.

There was considerable erosion of the furnace refractory by the highly acid slag, since no flux was used in the furnace charge. If further exploration of the deposit reveals that there is a considerable amount of nickel contained in the underlying serpentine, proper blending of earthy material with this more basic rock would partly solve the problem of rapid erosion of basic furnace lining.

Power and electrode consumption (88.1 kw.-hr. and 0.87 pound of graphite per pound of nickel produced, respectively) was not excessive considering the low grade of the ore, the small scale, and the short duration of the test. Power and electrode consumption would decrease in proportion to any increase in the grade of the ore.

The cobalt contained in the ore is partly recovered in the ferronickel product. The ferronickel contained approximately 0.25 percent cobalt, and the slag contained less than 0.02 percent. This cobalt probably would not be deleterious to the ferronickel for most commercial uses.

The results of the test indicate that it is technically feasible to recover a low-carbon ferronickel product from the Red Flats nickel ore. For a commercial operation a grade of ore considerably higher in nickel than the sample tested is indicated.

APPENDIX

Assay Logs of Drill Holes 1 to 23 (Excluding Hole 14)

Assays of Drill Hole No. 1 — Depth 60'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3	%Co
0–5	4–2	0.43	30.3			
5–10	4–3	.40				
10–15	4–4	.31				
15–20	4–5	.83	9.8	14.7	0.62	0.12
20–25	4–6	1.01	7.2			
25–30	4–7	.50	7.2			
30–35	4–8	.42				
35–40	4–9	.37				
40–45	4–10	.29				

Assays of Drill Hole No. 2 — Depth 50'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3
0–5	4–23	0.65			
5–10	4–24	.42			
10–15	4–25	.46			
15–20	5–1	.71			
20–25	5–2	.51			
25–30	5–3	.51			
30–35	5–4	.51			
35–40	5–5	.44			
40–45	5–6	.41			
45–50	5–7	.25			

Assays of Drill Hole No. 3 — Depth 60'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3	%Co
0–5'	3–15	0.52	26.6			
5–10'	3–16	.61				
10–15'	3–17	.52				
15–20'	3–18	.60				
20–25'	3–19	.50				
25–30'	3–20	.49				
30–35'	3–21	.91	8.8			
35–40'	3–22	1.26	10.6	14.9	0.62	0.04
40–45'	3–23	1.27	10.6			
45–50'	3–24	.88				
50–55'	3–25	.52				
55–60'	4–1	.51	7.5			

Assays of Drill Hole No. 4 — Depth 50'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3	%Co
0–5	2–20	1.13	45.0			
5–10	2–21	1.35	32.4	8.96	3.66	0.10
10–15	2–22	.50	14.3			
15–20	2–23	.30				
20–25	2–24	.83				
25–30	2–25	.55				
30–35	3–14	.51				
35–40	3–1	.73				
40–45	3–2	.56				
45–50	3–3	.84	6.4	7.58		

Assays of Drill Hole No. 5 — Depth 50'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3
0–5	6–21	0.19			
5–10	6–22	.67			
10–15	7–1	.45			
15–20	7–2	.62			
20–25	7–3	.68			
25–30	7–4	.56			
30–35	7–5	.39			
35–40	7–6	.24			
40–45	7–7	.22			
45–50	7–8	.21			

Assays of Drill Hole No. 6 — Depth 35'

Interval	Sample No.	%Ni	%Fe	%Mg	%CrO_3
0–5	8–25	0.43			
5–10	9–1	.61			
10–15	9–2	.61			
15–20	9–3	.71			
20–25	9–4	.51			
25–30	9–5	.64			
30–35	9–6	.42			

Composite sample of Hole 6 showed 0.06 % Hg

Assays of Drill Hole No. 7 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	2-10	0.51	31.0			
5-10	2-11	1.13	10.8		0.89	0.15
10-15	2-12	1.15	11.8			
15-20	2-13	1.16	10.0	15.2		
20-25	2-14	.81	7.8		.65	.15
25-30	2-15	.49	5.9			
30-35	2-16	.50				
35-40	2-17	.34				
40-45	2-18	1.87	5.4			.12
45-50	2-19	.20				

Assays of Drill Hole No. 8 — Depth 70'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	5-18	0.48			
5-10	5-19	.48			
10-15	5-20	.26			
15-20	5-21	.40			
20-25	5-22	.35			
25-30	5-23	.38			
30-35	5-24	.44			
35-40	5-25	.32	8.18		
40-45	6-1	.18			
45-50	6-2	.24			
50-55	6-3	.24			
55-60	6-4	.28			
60-65	6-5	.26			
65-70	6-6	.25			

Assays of Drill Hole No. 9 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	3-4	0.83	31.6			
5-10	3-5	1.35	25.8	15.7	1.66	0.07
10-15	3-6	1.13	6.4			
15-20	3-7	.84	12.2			
20-25	3-8	.43	6.4			
25-30	3-9	.38				
30-35	3-10	.27				
35-40	3-11	.25				
40-45	3-12	.61				
45-50	3-13	.50	17.4			

Assays of Drill Hole No. 10 — Depth 55'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	10-1	0.75	36.6			
5-10	10-2	1.24	35.5			
10-15	2-1	1.13	10.8		0.86	0.08
15-20	2-2	1.20	23.4	8.88		.08
20-25	2-3	.87	18.8			.09
25-30	2-4	.58				
30-35	2-5	.34				
35-40	2-6	.26				
40-45	2-7	.24				
45-50	2-8	.23				
50-55	2-9	.26				

Assays of Drill Hole No. 11 — Depth 50'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	9-8	0.33	25.6			
5-10	9-9	.67				
10-15	9-10	.64				
15-20	9-11	.97	19.4			
20-25	9-12	.95	11.0			
25-30	9-13	.84	10.4		0.74	
30-35	9-14	1.25	10.4	13.2		0.13
35-40	9-15	.97	8.8			
40-45	9-16	.59				
45-50	9-17	.58	7.6			

Assays of Drill Hole No. 17 — Depth 30'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	7-10	0.30	18.6		
5-10	7-11	1.35	18.8	7.32 (5-15)	0.02*
10-15	7-12	1.16	16.4		.07
15-20	7-13	.81	10.8		
20-25	7-14	.57			
25-30	7-15	.56	7.0		

* less than

Assays of Drill Hole No. 18 — Depth 25'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	7-17	0.48			
5-10	7-18	.52			
10-15	7-19	.45			
15-20	7-20	.41			
20-25	7-21	.42			

Assays of Drill Hole No. 19 — Depth 20'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	7-23	0.26			
5-10	7-24	.46			
10-15	7-25	.51			
15-20	8-1	.32			

Assays of Drill Hole No. 20 — Depth 40'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	8-3	0.98	41.4	6.47 (0-20)		
5-10	8-4	.83	41.6			
10-15	8-5	.98	35.0		1.05	0.11
15-20	8-6	1.22	20.2		.80	.10
20-25	8-7	.73	13.2			
25-30	8-8	.39				
30-35	8-9	.42				
35-40	8-10	.29	7.6			

1.00 % Ni (0-20)

Assays of Drill Hole No. 21 — Depth 20'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	8-20	0.62			
5-10	8-21	.48			
10-15	8-22	.41			
15-20	8-23	.51			

Assays of Drill Hole No. 22 — Depth 35'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3
0-5	8-12	0.23			
5-10	8-13	.23			
10-15	8-14	.25			
15-20	8-15	.29	6.9		
20-25	8-16	.23			
25-30	8-17	.23	6.05		
30-35	8-18	.22			

Assays of Drill Hole No. 23 — Depth 30'

Interval	Sample No.	% Ni	% Fe	% Mg	% CrO3	% Co
0-5	9-19	0.87	38.0	5.7	1.82	0.29
5-10	9-20	2.43	14.4	17.2 (5-30)	0.77 (5-30)	.17 (5-30)
10-15	9-21	1.07	10.6			
15-20	9-22	1.29	9.8			
20-25	9-23	1.16	10.0			
25-30	9-24	.51	6.6			

Composite sample of hole 23 is 0.01 % Hg

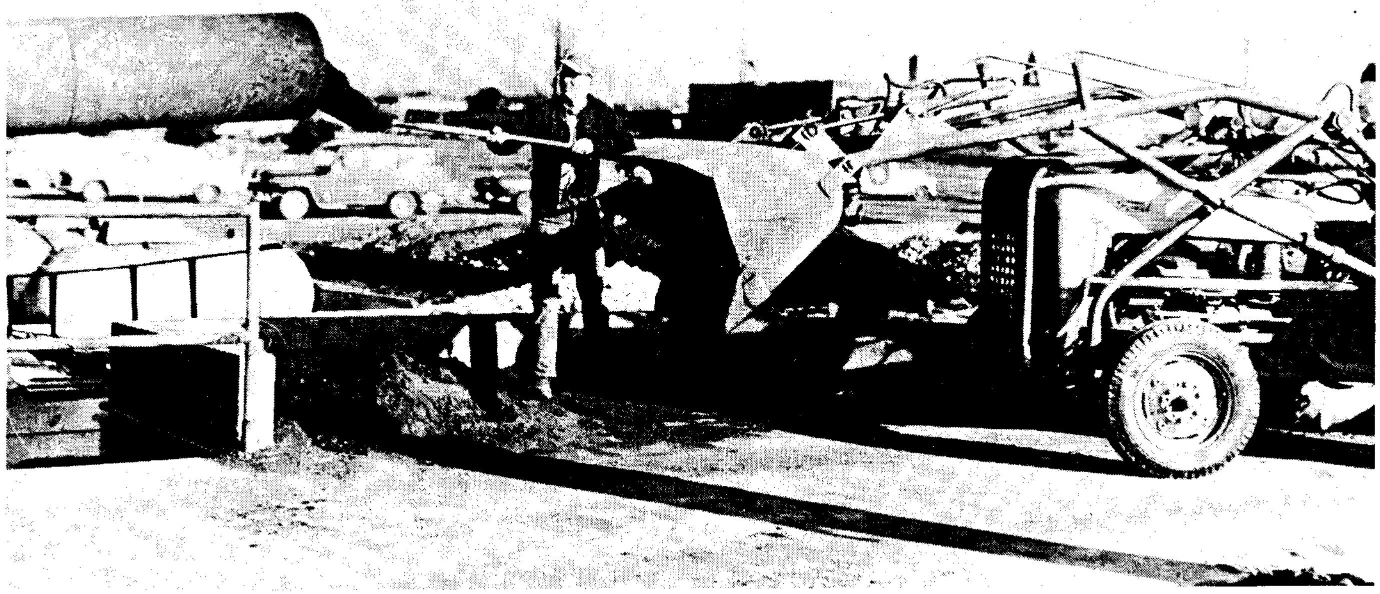


Figure 3. - Drying ore from Red Flats deposit.

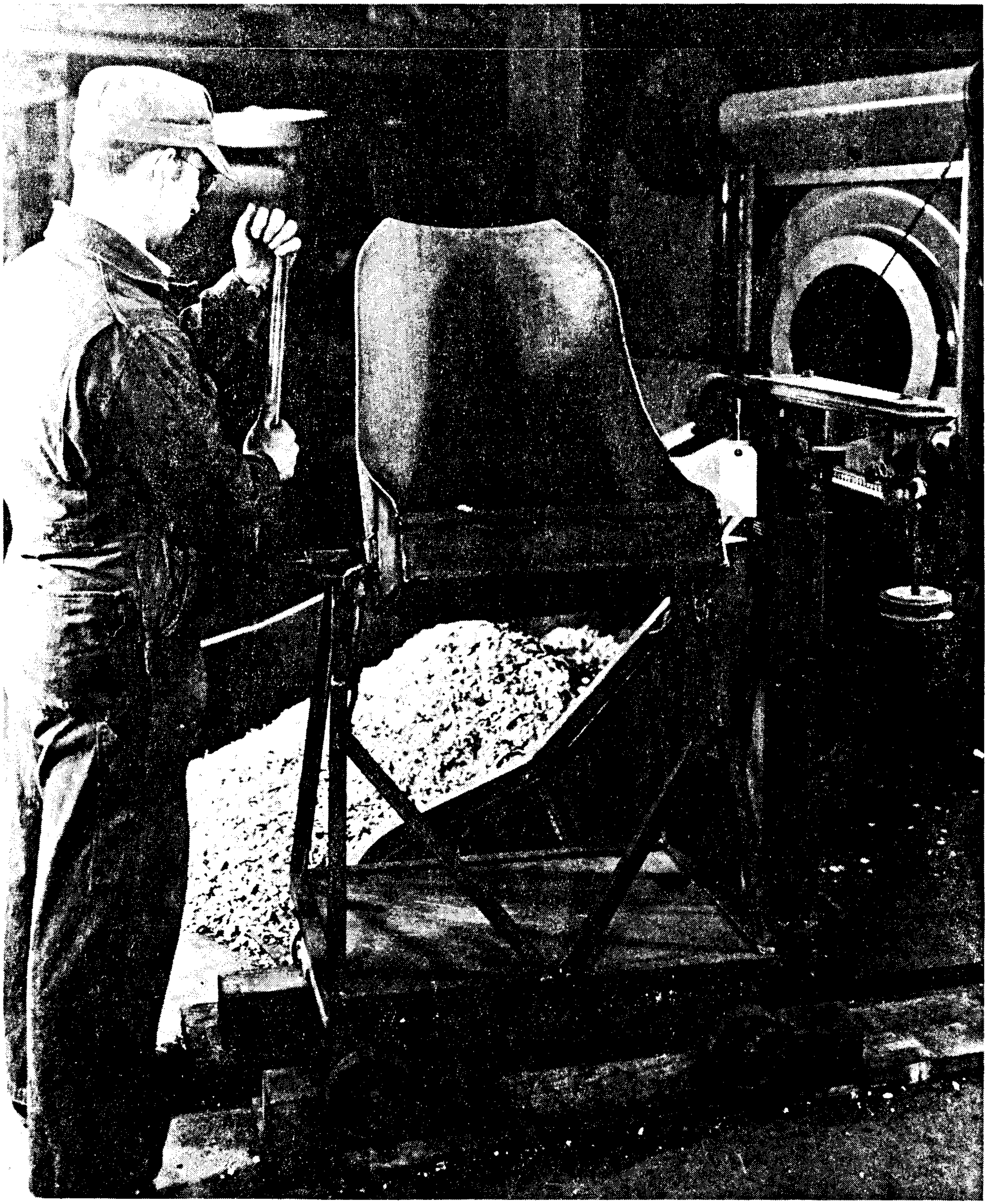


Figure 4. - Weighing the ore and reductant into charge mixer.

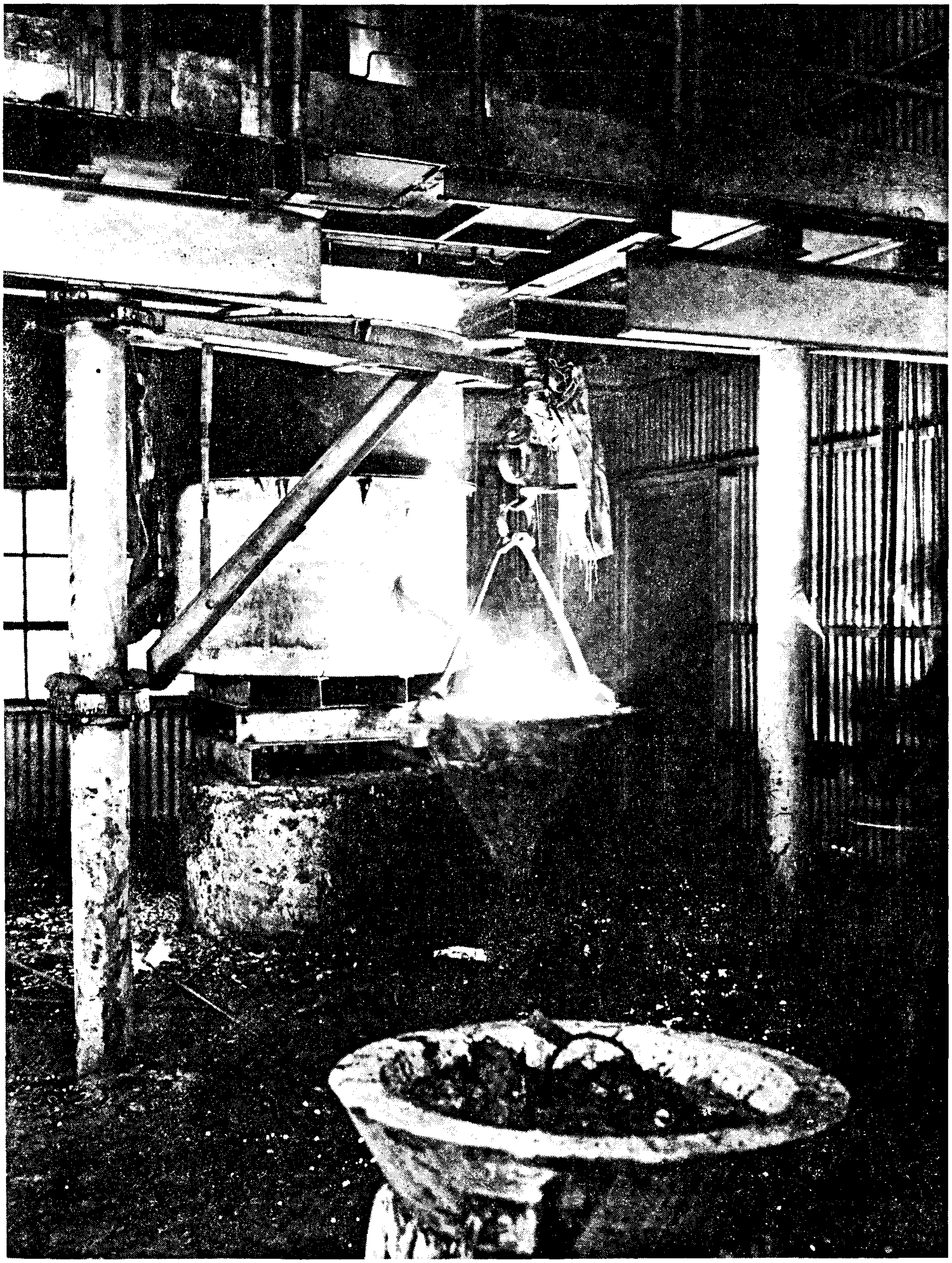


Figure 5. - Tapping slag from ESA furnace.

OREGON

Red Flats Quicksilver Mines, Gold Beach

Suggestions for a Comparative Test

Take a large sample, from hard rock from an area known to have shown quicksilver. Allow 50 lbs. in sample.

Pass through jaw crusher, all to pass 1/4" screen. Mix thoroughly on canvas or on flat smooth surface, lineoleum or oilcloth. Divide equally into 5 parts. Cut out a 5 lb. sample from each part. Number these T1, T2, T3, T4, T5, etc. Keep rejects and number each Tx1, Tx2, etc. Use strong paper bags for each sample.

T1. Send this sample to Smith Emery. Warn him that it may contain selenides of mercury. Ask for straight assay by standard methods of quicksilver and gold. Ask for a qualitative test on selenium.

T2. Clean grinder and all utensils thoroughly before making tests. Then grind dry to all pass 20 mesh. Put all product into retort, adding 2 lbs. clean charcoal crushed to pea size and 1 lb. lime ditto. Use proper clay to lute retort and proceed as usual. Keep retorted quicksilver in a properly numbered and tagged bottle, with date, weight, and calculated results. Take residue, put in bag and send to Abbot Hanks for gold assay.

T3. Grind all to 20 mesh, add water and a small amount of "soda" activator. Mix thoroughly for 10 or 15 minutes. Add considerable water in a large enamel pan (do not use any zinc tubs or pails). Then add small amount of depressant. Concentrate either using the small table or pans. Weigh any free mercury that can be saved. Dry and weigh concentrate, - retort concentrate, - send concentrate residue for gold assay to A. Hanks. Tag and weigh as before any drops of quicksilver that are obtained by retorting.

A comparison and analysis of the above three tests must first be made before proceeding further. If they all show quicksilver and gold content, even if in widely varying amounts, the following tests may then be made to throw additional light on the subject.

T4. - Neglect quicksilver - grind to 60 mesh and make bottle-agitation cyanide test for gold. With standard cut-out gold assay of heads and tails. Tag - record and compare.

T5. - Neglect gold - grind to 60 mesh and leach in 10% solution caustic soda - be very careful these solutions are extremely poisonous. Do not inhale fumes, do not get solution on skin.

There are various methods of precipitating the quicksilver, all difficult. Decant, wash and decant. Dry residue, mix thoroughly, cut out weighed sample and send for assay for gold.

A small lump from each of the original samples should be kept (before weighing) for microscopic examination.

With the results of the above tests on the same grade and type of ore we will be in a much better position to plan for mill-design tests and also for clues to interpret the nature of the deposit.

If these tests are carefully taken and made under conditions that allow of any independent observer and investigator repeating the same under his own control, they will be of real value - providing, of course, they show the expected "values".

John M. Nicol

METHODS OF SAMPLING

Because of the great magnitude of this property, the usual methods employed in sampling smaller deposits are self evidently not practical, if possible at all.

After considerable experimentation, it has been found that samples taken from the botton of pits ranging in depth from 4 - 10 ft, compare within a permissible degree of variation, with samples taken at the surface to a depth of 2 ft. It was noticed, however, that a sample taken from a vertical depth of 37 ft. and assayed for gold and mercury gave the result of 7 lbs. hg. and 0.4 oz Au. or $14.00 per ton, while a sample taken at the surface of this shaft and assayed for Hg. and Au. gave the result of Hg. 0.2% or 5 lbs. per ton and Au. .08 oz. or $2.80.

Just what significance we may attribute to this test is not known, as this was the only opportunity of taking a sample from such a depth.

For the above mentioned reason, we have sampled a broader area of the surface, rather than sinking pits to test at depth.

Samples are cut from ertical trenches, uniform in size, from the wall of the pits, ranging in depth from 2 - 6 ft. About 30 lbs. of material is taken for each sample. This material is crushed to minus ¼" and reduced by the usual quartering method, to about 4 lbs., which is ground until all will pass 80 mesh screen. This finely ground material is then well dried at a temperature, not exceeding 105 degrees F. and thoroughly mixed.

METHODS OF TESTING

Owing to the fact that an average of 17 elements occur in this ore, it has been found necessary to make a number of changes in the usual method of assaying.

For instance, the most commonly employed method of testing for mercury is the Whitton test. This test has been found entirely unsatisfactory.

We believe this is primarily due to the fact, that the ore is impregnated with a bituminous substance, tentatively judged to be claterite. This material volatilizes at about the same temperature as the Hg. content of the ore. The oily vapor produced by the volatilization of the bitumens, will invariably condense on the provided gold leaf, preventing amalgamation of the Hg. vapors.

Another condition believed to be deleterious to the Whitton test, is the fact that the ore is basically a hydrated magnesian silicate.

Upon application of the heat, sufficient pressure is generated from the water of hydration of the ore to cause leakage of the Hg. vapors from the ignition chamber of the Whitton, thus rendering low or negative results.

It has been found that a well mixed and dried sample will always give higher results by retorting, in the ordinary pot retort with water jacketed condenser pipe, than with the Whitton test.

TESTING FOR GOLD

A large portion of the gold content of this ore occurs in its free or native state, and it is our estimate that at least 90% would pass a 200 mesh screen as it occurs in the ore.

For this reason and as already stated, because of the interfering elements in the ore, it has been found necessary to make slight deviations from the usual methods of fluxing for fusion.

The analysis of this ore reveals, among other things, a cromium oxide content ranging from 1% to 12% magnesium oxide 2% to 10%, chrystolite asbestos, 4% to 8%, hydrated magnesium silicates other than asbestos, 10% to 30%. It is a well recognized fact, that these substances are fused only with great difficulty, if at all.

Since in the theory of fire assaying the finely ground ore is mixed with litherge and other suitable materials for reducing the fusion temperature of the elements in the ore. A large portion of the litharge being reduced to small globules of metallic lead as the result of the reducing power of the ore and to collect the particles of precious metals as it settles through the molten slag to the bottom of the crucible.

If the ore is of such a nature, that the litharge will melt and settle to the bottom before the slag of the ore has become sufficiently viscous, then obviously the result of the assay will be low or nil, regardless of the metaliferous content of the ore.

After considerable experimentation, we have been able to overcome this condition by application of the following formula:

Ore minus 80 mes	0.5 Assay ton
Sodium bicarbonate	25 Grams
Potassium Carbonate	20 Grams
Litharge Silver free	58.332 "

Si O C. P.	25 Grams
Borax	5 "
Flour	1.5 "
C. P. Silver foil	5 Milligrams

Mix this charge thoroughly and place in 30 gram crucible, tap gently to settle. Over the top of the charge, place one assay ton of litharge and cover tich layer of sodium chloride ¼" thick. Place crucible in cold furnace and bring to fusion temperature, gradually, being careful to avoid voilent escaping of gasses at first of fusion. This process should require between 45 min. and 1 hour, and should be held at the highest temperature obtainable in the assay furnace for the last part of the fusion. When quiet and viscous, pour into mould, break slag from lead botton which should weigh 18 to 20 grams, cupell and calculate in the usual manner, 1 Mi Au. recovered from ½ assay ton being equal to 2 oz. Au. per ton of ore.

It is not infrequent to find an alloy of Platinum and the platinum group of metals in the cupell. This will manifest itself in one of two ways. First the bead at the finish of the cupellation, will refuse to dissolve in HNO. Second, the bead will burst and assume a caulifour like appearance. In either case it has been found advisable to add 12 times the weight of the bead, of C. P. Silver foil. Mix with test lead and scorify. This is based on the fact that Platinum alloyed with 12 times its weight of silver is readily soluable in HNO.

Note, the Platinum group of metals are found throughout the ore body varying in amounts from 0.01 oz. per ton to over 1 oz. per ton. It is interesting to notice that when the gold content is below the average the Platinum content will be correspondingly higher.

Another interesting phenomena is the fact that the Hg. content in lbs. per ton will compare quite closely with the Au. value in dollars and cents per ton.

STATEMENT REGARDING SAMPLES TAKEN AND THEIR VALUES

As already stated, a plan of systematic mapped sampling is being carried out. Since this work is not completed, no attempt will be made to give a detailed account of the results of this work until it is completed, which should be about the first of August. Instead it is believed sufficient to state what over 100 samples have recently been cut at random from widely separated points all over the area of the property in order to establish uniformity and consistency of the values in the deposit.

From the 100 samples, there were no blanks; each sample showing at least traces of both Hg. and Au. The average of the samples taken were over 4 lbs. Hg. per ton and approximately 0.1 oz. Au. per ton. It is to be remembered that these samples were taken at random over an area 1 mile wide and 2 miles long.

Another example, 160 acres in the approximate center of the property was mapped out and 32 shallow pits from 2 tl 4' in depth were sunk in the approximate center of each 5 acres of the 160 acres. The average assay value recovered from these samples, was 7.8 lbs, Hg. per ton. $10.76 per ton in Au.

Note these pits are marked and the work can be duplicated with approximately the same results.

From the sampling work so far completed, we find that an average from any two samples taken at random within the boundaries of the property, will give in excess of 4 lbs. Hg. and $3.00 Au.

METHOD OF MINING

Because of the extremely friable nature of the ore, and considering the topography of the property, the most efficient and economical means of mining will be the open pit method. It is our judgment that very little blasting will be necessary.

The most economical means of conveying the ore to the coarse ore bin, which suggests itself, is a track type tractor and powered self-loading scraper of the type generally called carry-all.

A survey of similar earth and rock moving projects has led us to believe this material can be charged into the coarse ore bin for a total cost, not to exceed 20¢ per ton of ore.

SUGGESTED MILL FLOW SHEET

Coarse ore bin with shaker screen designed to by-pass all material minus ¼" to fine ore bin. A steel rail grizzly provided to hold for hand sledging, all boulders larger than dimention of jaw crusher. A mechanical feeder from the coarse ore bin into a Kue Ken double acting jaw crusher with the desired capacity of mine run to minus ¼". The crushed ore discharging into fine ore bin of desired capacity. The crushed ore fed by adjustable mechanical feeder into standard make of ball mill designed for semi-fine grinding. The ball mill operating in closed circuit with Dorr rake type classifier, adjusted to discharge minus 20 mesh product into conditioning tank with same capacity of pulp as the capacity of grinding equipment, unclassified product or oversize being returned to feed of mill.

It has been found advisable to condition the classified pulp in an alkaline solution, to digest the bitumens in the pulp, to lower surface tention of pulp, to facilitate coalesence of particle of free mercury.

Three tanks with pulp capacity equal to that of the grinding units, arranged where they will be charged by gravity from the classifier. These tanks will be equipped with mechanical agitators designed to hold the pulp in suspension. Each tank when charged, will be allowed to condition for approximately 24 hours. After the first tank is charged, the pulp flow will be diverted to the second tank. While the third tank is being charged, the first tank after conditioning for 24 hrs. is discharged over wet gravity concentrators, where a concentrate at a ratio varying with the ore, of from 10 to 1 or 20 to 1 is made.

This concentrate is then dried and retorted. The calcined tailings from the retort will then be charged into suitably arranged cyanide tanks, where the Au. content will be recovered by direct leaching.

Note, as nearly as could be determined by labratory test, a slight increase in recovery could be realized by separating the slimes from the sand before passing conditioned pulp over concentrating tables. This additional recovery does not, however, seem to justify the added cost of installing and operating thickeners and agitators.

At this stage of the above described processing, as nearly as we have been able to determine, by small scale testing, about 70% of the existing Hg. and Au. content will have been recovered.

An assay of the material discharged from the leaching tanks will reveal from 20 to 28% nickel and from 1 oz. to 2 oz. of metals of the platinum group. This material will then be reconcentrated at an undetermined ratio and shipped to a smelter for refining.

Improvements in the above outlined flow sheet will no doubt manifest themselves as our work progresses.

All values alluded to as assay values have been confirmed by mill test on a semi-commercial scale.

John A. Shoff

Carl Smedberg

GENERAL REPORT

RED FLATS GOLD MINE

SOUTH MCKINLEY MINING DISTRICT

CURRY COUNTY, OREGON

William F. Hayden
Engineer
Continental Bank Building
Salt Lake City

1932.

Dennis A. Winn

Section 3.

Report of Dana W. Bowers, Assayer.

(C O P Y)

GOLD HILL, Ore. Apr. 3, 1939

Report of Assays and ore treatment of Red Flats ore for Phil. R. Adams and Associates, Gold Beach, Oregon.

# 254-	Ore samples from Red Flats, Oreg.						
#1-	Fire Assay- Gold	a trace					
	Amal. " "	.11 oz.	per T.	value $	3.85	per	T.
#2	Fire " "	a trace					
	Amal. " "	.15 oz.	per T.	"	5.25	"	T.
#8	Fire " "	a trace					
	Amal. " "	.1 oz.	" T.	"	3.50	"	T.
#9	Fire " "	a trace					
	Amal. " "	.06 oz.	" T.	"	2.10	"	T.
#10	Fire " "	a trace					
	Amal. " "	.09 oz.	" T.	"	3.15	"	T.

#s1-A, 2 comp.-3, 4, 5, 6, 7, 11, 12, 14, 15,
composite sample.
Fire Assay- Gold a trace
Amal. " " .07 oz. per T. value 2.45 per T.
Amal. " " .0966 oz. " T. Average3.38 " T.

- -

255. Sample "Pyritic ore S. of Red Flats"
Fire Assay -- Gold -- .15 oz per. T. Value 5.25 per T.

¢¢¢

256. Amal. Mill Test of all Red Flats samples - 25 lbs.
Gold recovered, .043 oz., per T. value $1.50

¢¢¢

257. Flotation Test of Red Flats ore, composite sample, 20 lbs.
Extraction per T. gold, .0033 oz. value, ...$ .115 per T.

¢¢¢

258. Cyanide Test of all samples of Red Flats ore-
Gold- .12 oz. per T. ... value ... ... $ 4.20 per T.
Silver .8 " " " ... " ... ... .33 " "

¢¢¢

Amenable to a special Cyanidation treatment for extraction of gold and silver.

The quicksilver content of this ore is not of economic value, if based upon the composite sample, except as incidental to cyanidation.

(SIGNED) Dana W. Bowers,
ASSAYER.

GENERAL REPORT

RED FLATS GOLD MINE

SOUTH MCKINLEY MINING DISTRICT

Curry County, Oregon.

ACKNOWLEDGEMENTS:

We are greatly indebted to--H. P. Bostaph, Chemical and Mechanical Engineer of Fort Worth, Texas, for drawing maps, etc., together with general information.

To E. Derwent, Chemical and Metallurgical Engineer, of Pittsburg, Pennsylvania and Eureka, California, for assays and general information affecting mill installation.

Some data was used from Government bulletins and maps.

To Carl Smedburg and Alford Lachance of Gold Beach, Oregon, for conducting us over the property and for many courtesies extended during the several visits to the property while examinations were being made.

ooOoo

GENERAL REPORT

RED FLATS GOLD MINE

SOUTH MCKINLEY MINING DISTRICT

Curry County, Oregon

PROPERTY:

The property which is the subject of this report embraces thirty-six mining claims or seven hundred and twenty acres named as follows:

Cold Springs	No. 1	)	
Cold Springs	No. 2	)	
Cold Springs	No. 3	)	
Cold Springs	No. 4	)	
Gravel Gold Assn.	No. 1	)	
Gravel Gold Assn.	No. 2	)	
Gravel Gold Assn.	No. 3	)	
Gravel Gold Assn.	No. 4	)	
Reporter	No. 1	)	
Reporter	No. 2	)	
Reporter	No. 3	)	
Reporter	No. 4	)	UNPATENTED
Ocean View Assn.	No. 1	)	
Ocean View Assn.	No. 2	)	
Ocean View Assn.	No. 3	)	
Ocean View Assn.	No. 4	)	
Ocean View Assn.	No. 5	)	
Ocean View Assn.	No. 6	)	
Ocean View Assn.	No. 7	)	
Ocean View Assn.	No. 8	)	
Red Elk	No. 1	)	
Red Elk	No. 2	)	
Red Elk	No. 3	)	
Red Elk	No. 4	)	

Red Flats Assn.	No. 1	)	
Red Flats Assn.	No. 2	)	
Red Flats Assn.	No. 3	)	
Red Flats Assn.	No. 4	)	
Red Flats Assn.	No. 5	)	UNPATENTED
Red Flats Assn.	No. 6	)	
Red Flats Assn.	No. 7	)	
Red Flats Assn.	No. 8	)	

LOCATION:

The Red Flats property is located in what would be termed the South McKinley or Pyramid Rock Mining District in Township 37 South, Range 13 West of Willamette Meridian, Curry County, Oregon, approximately seven miles easterly from the Pacific Ocean tide water and the Oregon Coast (Roosevelt) Highway which is the military highway of the Pacific Coast. The nearest town is Gold Beach, Oregon, located at the mouth of Rogue River on the Oregon Coast (Roosevelt) Highway, a distance of approximately nine miles from the property. The village of Gold Beach is the county seat of Curry County, Oregon and maintains large mercantile establishments where supplies of all kinds for the mine may be had.

A map of Curry County is included in the compilation of this report.

TITLES:

All the claims described herein are unpatented and are held under the provisions of the Mining Laws of the United States and the State of Oregon governing the location of mining claims. The titles to the claims are unincumbered and an abstract will be furnished fi desired but the title is clear.

CLIMATE:

The climatic conditions would be regarded as exceptional in that there is practically no snow during the winter months to interfere with continuous operations. The elevation at the mine over the different areas is 2140 to 2260 feet above the sea level.

MAIL:

Mail to the mine would be delivered to the Post Office nearest the mine.

WATER:

There are two streams of water which may be used either for camp or mill purposes adjacent to the property. These streams are found to the north and west of the property but are not noted on the map which accompanies this report. The flow of water in these streams is not sufficiently large to accommodate any size mill production but we believe it would be sufficient to supply pure water to the camp.

Water may be had from Pistol River immediately East of the property and from Hunter's Creek which is slightly northwest of the property. Both of the streams are in close proximity to the mineralized areas of the Red Flats property and either of the streams would carry sufficient water to accommodate large mill production.

TOPOGRAPHY:

The Red Flats property is located on what would be called the Eastern slope of the Coast Mountain Range about one-half mile from the Hunter's Creek stream level and an equal distance from the floor of the Pistol River gorge.

The topography of the areas under consideration would be described as rolling to flat or level country. At different sections of the property will be found some swales or small gulleys. The elevation is 2140 to 2260 feet above sea level--this elevation is attained in a distance of seven miles from the ocean permitting the construction of either roads or railroads with an approximate grade of 300 feet to the mile. The property should be considered as easily accessible.

ROADS AND TRANSPORTATION:

As indicated under topography the property is accessible in that a road or railroad can be constructed either along the shores of Hunter's Creek or Pistol River. (See map which clearly sets out the different routes.)

The road parallelling Hunter's Creek would be a distance of 7½ miles and the road up Pistol River would be something over 15 miles in length to the property. Both streams empty into the Pacific Ocean. Hunter's Creek two miles and Pistol River Twelve miles south of Gold Beach.

The Hunter's Creek route would be somewhat higher in cost per mile but is shorter and would tap not only the product of the mine but would accommodate the transportation of millions of feet of timber from the sections it would reach.

Likewise the railroad or road constructed along Pistol River.

It is estimated that there is in excess of one billion feet of timber in the Hunter's Creek and Pistol River Basins which could be brought to ocean transportation if a road or railroad were constructed upo the course of either of these streams.

The question of transportation in this case is not a serious one since the character of values recovered at this mine is gold and the only necessary transportation, therefore, of any consequence would be to ship supplies into the mine from the highway. The question of delivering the gold to the mint is insignificant. There is no transportation problem nor must gold depend upon the rise and fall of markets. One of the pleasing facts in connection with the operation of a gold mine.

The fact does remain, however, that the gold values of this mine are carried in what is termed iron ores containing about thirty-six percent brown hematite or iron oxide and about twelve percent manganese. This constitutes a commercial iron ore and in this case a railroad should be constructed to accommodate shipment of this product to tide water after the gold has been extracted.

Transportation problems affecting the shipment of ore from the mine or timber over a railroad constructed to the mine is one of considerable satisfaction since the products would be loaded upon ocean going vessels at a nearby port.

Our chief construction engineer has just completed a survey of the road and railroad grades and conditions affecting same from tide water to the properties, and his estimate of road construction carrying sufficient grade and structure to enable heavy trucking would cost approximately two thousand dollars per mile. If a lower percentage of grade would be required to facilitate more rapid transportation the cost for some of the miles would be up to three thousand dollars. Railroad construction on either Hunter's Creek or Pistol River he estimates would cost ten to twelve thousand dollars per mile for complete railroad construction from tide water to the mine.

TELEPHONE:

Telephone connections made through the Gold Beach Exchange of the West Coast Telephone Company is now in close proximity to the property and can be extended to the mine at a very small expense.

EQUIPMENT:

There is no equipment or buildings upon the property at the present time.

HISTORY:

The property has been known for many years, according to the best information obtainable, to carry gold and platinum. Rich strickes of gold in the adjacent country to this property have been reported from time to time and gold ahs been known to exist in what is called the Red Flats for many years, as well as in the formation underlying this property. The grade of ore is such that operations upon a large scale would be required in order to be profitable.

INDUSTRY:

The industry near the property is confined to grazing and sheep and cattle raising with little other land developed.

TIMBER:

Mr. Otto Ismert of Pistol River who bears a fine reputation as a cruiser having done work for both Curry County, and State of Oregon in this line and whose judgement we regard as excellent gives us the following figures and information;

A railroad constructed upon the course of Hunter's Creek for seven miles to the mine would tap énroute and in the surrounding areas five hundred million feet of timber; fifteen percent of which would be Port Orford White Cedar and eighty-five percent fir (Oregon Pine), with one hundred twenty-five million feet of tan oak and other merchantable timber.

If a road or railroad were constructed up Pistol River fifteen miles it would pass through or tap eight hundred million feet of timber--five percent of which would be Port Orford White Cedar and ninety-five of fir (Oregon Pine) with an additional two hundred million feet of tan oak and other merchantable timber.

GEOLOGY AND ORE DEPOSITION:

The geology of the area covered by the Red Flats Gold Mining property is unquestionably formations of igneous origin. The particular areas under discussion having a topography as above described consisting of rolling, low-lying hills, grading almost to level areas.

The best authorities are somewhat at a loss to understand the exact geology and ore deposition of this deposit. It appears the iron stained material carrying the gold, etc. is a breaking down of what was a great intrusion, since the underlying strate of the ore mass is entirely of igneous origin and may be classed or determined as an eruptive beccia resembling a lavatic character of rock which may at depth be classified as an adecite or diorite.

The underlying strata or rock which immediately underlies the ore mass carries 19 to 20 % ferric oxide and assays 40¢ per ton in gold. This material evidently having been eroded and the surface exposed became oxidized, caused by slow deteriation of the lavas due to surface waters, formed what is now the large mineralized area called the Red Flats. The original material was evidently an altered andecite or diorite.

The gold is accounted for, in that the underlying strata, according to assays, carried gold values.

It is evidenced over the areas above described that fragmental and small quartz crystals impregnate the deposit, which may constitute or be one of the gold bearing agencies of these deposits. In the breaking down of the mass which now constitutes the mineralized deposits we find protrusions of the under structure exposed over the different sections of the mineralized areas, which lie horizontally. While these protrusions which are part of the intrusive under structure appear frequently upon the surface, the number of same would not be sufficient to cause a replacement of more than thirty percent of the mass until a reasonable depth is attained. The ore mass which then closely examined, is found to be composed of principally brown hematite or oxide of iron (probably hydrated) in which quartz is a constituent, to a known tested thickness of seven to eight feet extending over the entire property, except where the underlying rock is exposed at different intervals upon the surface as above described. No sedimentary rock is found. We believe the deposit would not be classed entirely as gossan.

When the original examination of the property was made we found a large quartz ledge to the South and slightly East of the Red Flats deposit. A development consisting of an open cut upon the ledge at a point twenty five hundred feet South and East of the Red Flats deposit we found the ledge prominently exposed carrying a width in excess of twenty feet across its face. The eastern portion of the ledge somewhat stained with iron oxide but the Western exposure fo the deposit is found to be of a silvery white or grayish colored quartz, apparently embedded in a Schistose material.

The contact to the East, we believe, would be classed as schist; to the West the contact is slate which in turn is flanked by a great porphyry dike. The strike of the ledge is North 20 degrees West. The dip of the formation with the amount of work that has been done makes it difficult to determine, but in so far as we could see, would be about 65 degrees East, slightly North.

Assays of this quatrtz by Derwent gave $4.60 per ton in gold and platinum tests by Black & Deason, Salt Lake Chemists, gave a return of 90¢ per ton in platinum. The samples were taken close to the surface and it is theopinion of the writer that when depth is attained this ore may develop good commercial values in platinum from ore in place.

The ledge is exposed upon the surface to the South or the open cut described for a distance of probably six to eight hundred feet and also is observed as a cropping upon the surface as it strikes North from this point for a distance of some twenty-five hundred feet where it is lost in the Red Flats ore deposit.

VALUES:

The values contained in the ore found upon this property consist of gold, platinum, iron oxide and manganese. The manganese coming in the form of a manganese dioxide. The iron content is 36.45% and the dioxide is 11.72%, which totals 48.17% mineral content.

The following assays show the values contained in the ores of this property.

Eureka, California
July 22, 1930

Mr. W. F. Hayden, E. M.
Gold Beach, Oregon

The sample of Ore marked Red Flats contained the following:

Incoluable Residue	21.34%
Iron Oxide	36.45%
Alumina	9.40%
Manganese Dioxide	11.72%
Lime	1.75%
Magnesia	0.50%
Loss on Ignition	14.40%

E. Derwent
An & Met Chem.

Our Metallurgist informs us that in his opinion the silisa content in this ore under regular treatment will probably be greatly reduced.

Eureka, California — and gold price
August 18, 1930

W. F. Hayden, E. M.
Gold Beach, Oregon

Red Flats.

The samples of Ore Marked 1 to 24 and 25 to 50 were Leached with a solution of Cyanide having a percentage of 0.20 for seven days and the Gold recovered.

Sample 1-24

Heads.	Gold 0.085		$1.70 per ton of ore.
Cyanide Leach.	Gold 0.0244	Grams	$1.60 per ton of ore.
Tailings.	Gold Trace		

Twenty Pounds of Ore treated indicating a Gold recovery of ninety-five percent. (95.00%)

Sample 25.-50.

Heads.	Gold 0.11 ozs.	$2.20 per ton of ore.
Cyanide Leach .	Gold 0.04 Grams	$2.10 per ton of ore.
Tailings.	Gold0.003 ozs.	$0.06 per ton of ore.

Twenty Pounds of Ore treated indicating a Gold recovery of ninety-five point four percent. (95.40%)

E. Derwent
Chem & Met Eng.

ASSAYS:

The above assays comprise both a fire assay and a leaching test of some fifty samples taken over a co-ordinate system of sampling of the property. The samples taken by the use of a "fishtail" drill to a depth of seven to eight feet over the entire property except where the underlying rick extends through and is exposed upon the surface, the areas of which would be small.

The tests for platinum made by Black & Deason of Salt Lake City, a very reliable firm of chemists, give a return of 45¢ per ton in platinum. A second leaching test was made by the use of a leaching process devised by Summit Markesbery, of Coos County, Oregon, and the result of this leaching test was a recovery of $2.80 per ton in gold and 60¢ per ton in platinum. (See plate marked "Exhibits.")

TREATMENT PLANT:

The question of treatment of the ores of the Red Flats property while nor difficult in any one respect at the same time every precaution must be used in order to determine which would be the most economical treatment for these ores as there are several standard methods commonly applied to the treatment fo such ores. Several tests affecting the recovery of the values contained in the ore have been made. The first test or assay was the regular fire test resulting in a recovery of gold and platinum, as the above assays on pages 8 to 10 inclusive will show.

To the concentration plant or to the mill or grinding unit employed, should be added a regular cyaniding unit for the treatment of the tailings to recover the gold which has not been saved by table concentration. The iron oxide and manganese which constiutes the bulk of the tailings treated as they pass over the concentrate table will not be injured by the solution and will be saved and so sold as commercial iron ore.

LEACHING TEST: A complete leaching test by cyanidation was made in the Derwent laboratory. The ore used was a composite sample of the larger areas of the property. The samples here were obtained by the use of fishtail drills so that the ore content was equalized from the surface through the depth tested seven to eight feet. We are reasonable sure, therefore, that a uniform sample was obtained. The treatment applied in the leaching of this ore was a weak solution (2 lbs. of cyanide) to the ton.

OPERATION AND REQUIREMENTS:

The following is approximate construction requirements of the mill and mine equipment together with mine operation data which is largely an approximation.

The cost of mill operation under the circumstances here is undertain and difficult to estimate but it is the opinion of the writer that a 500 ton Ball Mill installation will accommodate a daily capacity in excess of its regular estimated daily capacity of tons of the class of ore that is found in this mine, for the reason that there is not a large percentage of coarse material necessary to be ground obviating the necessity of extensive primary grinding, which evidences here an economic quality making it possible to treat a large tonnage of ore at a reasonable cost. We would, therefore, feel somewhat inclined to italicize the economic situation developed here.

The examination of the Red Flats property indicates an exceptional tonnage of what must be termed positive ores and because of this fact leaves no alternative to the writer but to recommend development of this mine though the per ton value in gold is low. This opinion is based entirely upon the large ore reserves and assay values found.

Consideration, however, must be given here to the fact that the ore which is to be treated will carry approximately forty percent iron oxide and manganese content which we believe may be recovered and saved for shipment. While this is not a new experiment, the writer would prefer to include same as prospective profits only, wishes to reserve the right to place th recovery or gross profit at less than one-half of the price quoted for this class of ore because of transportation and the uncertainty of the market for these products in this district.

The location of the mill for the treatment of the ores of the Red Flats property will be about one-half mile down the mountain at an elevation of approximately five hundred feet above sea level which will necessitate the ore being conveyed from the 2100 foot level to a 500 foot level unless discovered to extend to lower levels which is probable. The question of economic construction for the delivery of the ore from mine to mill is one of considerable moment. Our recommendations will include a surface tram installation, and our estimate for the cost of equipment and installation would be $10,000.00.

<u>POWER:</u>

One of the essentials affecting the profit of a large mining operation such as would be conducted here is the power cost.

Our first consideration is the installation of a 300 H. P. Hydro-electric plant which may be installed upon the Pistol River at a distance from the mill and mine of about five miles, necessitating the installation of high tension wires over this distance. An estimated cost of this installation when completed including freight would be approximately $40,000.00.

To furnish 300 H.P. by the use of Diesel engines would cost approximately $20,000.00; the difference in the first instance being that no fuel would be required while in the second instance the installation is cheaper but it would be necessary to find a continuous supply of distillate.

MINING COSTS:

The ores of the Red Flats property may be mined very cheaply and it is recommended that a 1 yard capacity gasoline or electric shobel be employed which will convey the ore to a surface tram which will in turn deliver the ore from the surface pit to the mill.

While it is believed that the ore can be delivered to the mill at 40¢ per ton, for the reason that the ore lies practically as horizontal deposits from the surface breasts of seven to eight feet, we are estimating the cost to be 60¢ per ton, as a matter of safety.

MILL TREATMENT:

After careful consideration, we find that even though a large percentage of the ores to be treated are soft and fine and contain only a reasonable percentage of material which will require primary grinding, fine grinding to possibly 150 mesh will be necessary to recover the values from these ores. We place the mill treatment cost, the saving of the iron ores and the cyaniding of the tailings at a total cost of $1.00 per ton, placing, therefore, the total of the mining and milling cost at $1.60 per ton. It is believed that this cost may be materially reduced but for safety we place the estimate as above.

MILL COST AND MINING EQUIPMENT:

Construction, installation ready to place mill in operation, we would place at a total of

		$108,535.00	$108,535.00
Mine:			
Gasoline Shovel	8,000.00		
Tram	10,000.00		
Road Construction(7½mi.)	15,000.00		
Automotive Equipment	10,000.00		
	43,000.00		
Miscellaneous	25,000.00	68,000.00	68,000,00
If Diesel power used		20,000.00	
If Hydro-electric power used			40,000.00
		$196,535.00	$216,535.00

The above prices have been checked to the best of our ability but are given here as an estimate or approximate cost of the equipment above outlined.

ORE RESERVES:

The situation such as developed at the Red Flats Gold Mine property is exceptional in that practically the entire ore body is exposed to daylight. The burden of proof, therefore, rests in obtaining the definite values of the ores from the different sections of the property wich permits an estimate of what would be termed positive ore.

The following is an estimate taken f om the evidence produced by the use of fishtail drills over the areas described in this report which include the estimating of tonnage found upon the eight hundred acres of the property and the value of the ore as shown by the assays, mining and milling costs and net profits at the mine.

200 acres 8 feet deep at 160#	per cu. ft.	5,575,680 tons
600 acres 7 feet deep at 160#	per cu. ft.	14,136,160 tons
	Total	19,711,840 tons
Less 30% boulders and rocks		5, 913,552
	Total (Net)	13,798,288 tons
Sell		
Platinum & Gold 13,798,288 tons@ $2.20		$30,356,433
Minign & Milling Cost, 13,798,288 tons @ $1.60		22,077,260
	Total	$ 8,279,173

This estim te does not include overhead, depreciation, interest upon monies invested nor the miscellaneous requirements usually occurring at mines of this type.

The above estimates include only the amount of ore actually tested by the use of a fishtail drill to a depth of seven to eight feet together with the assay results as recorded. The potentialities of the mine over and above that which may be classed as positive ores would be very great.

RECOMMENDATIONS: It is recommended that the Red Flats property be intensely developed to substantiate the work already doen and verify assay results. We would then recommend that a gravity flow concentrating plant employing a Ball Mill, tables, amalgamators, etc., be used; that the tailings be cyanided in order to save the gold which cannot be saved by gravity concentration.

The recommendations for operation are based solely upon the large tonnage of ore available and upon the assay values and tests of the ores of the Red Flats Gold Deposits which have been made.

The writer believes that fifty shafts to a depth of seven or eight feet be sunk or drilling be done co-ordinately over the ore deposits of this property and that leaching tests be made from a mixed sampling of the ores of these shafts to corroborate the "fishtail" sampling already done. In making this recommendation, it is the writers belief that the assay values will check closely with those already made.

That this work be done forthwish.

Our further recomendations would be to construct a road from tide water or the Roosevelt Highway to the mine, a distance of approximately seven and one-half miles and that if deemed wise after the mine is placed in operation a railroad be constructed for delivery of the possible iron ores recovered and to accommodate the large timber interests of the section in which the mine is located, making the railroad a profitable investment aside from its function of ore deliver.

REVIEW AND CONCLUSIONS:

The power requirements in connection with the operation of this property will be furnished by the use of oil burning and Diesel engines. Freight from San Francisco would be delivered by ocean going vessels to Brookings and nearby port, and the charges for distillate would not be excessive.

TITLES: The property is free of all incumbrances except the paramount title of the United States Government governing the locations of mining claims and the obligations to the state of Oregon in which the mining claims are located. This is the best class of title since the development of the mine pays the assessment upon the claims and no taxes are exacted except upon the recoveries. The Government looks with favor upon gold mining.

The water situation would be regarded as fair.

The topography are fully covered in this report.

The geology and ore deposition is favorable.

The ore reserves and potentialities, we believe, would be considered as exceptional.

The cost of the installation in order to place the mine in operation in comparison with the positive ores in sight; together with the mine's potentialities would be considered reasonable.

The economic natural resources are good.

No unfavorable climatic conditions affecting continuous operation.

CONCLUSION:

The accessibility of the mine, the amount of positive ore available, together with the cheap mining costs invite a dicision that this mine under-go intensive development, however, we would recommend that development work to the extent of shallow shafts to the depth of eight to twelve or fifteen feet be sunk, one or two upon each claim; a composite sample taken from the ore of each shaft and the same be leached for gold content so that it may be compared with the ores already tested. When this is done and if the assays check with those already made, and we believe they will, it is our opinion that the equipping of this mine as above outlined would be warranted, though all of the ore would be considered as low grade.

With this done and the metallurgy of the ores determined, as seems has been done in this case, the possibilities are such that hte expenditures upon the property for equipment, etc., may be safely entailed to the end that the ores of this mine may be mined and treated.

Respectfully,

WILLIAM F. HAYDEN.

SUPPLEMENTAL REPORT

RED FLATS GOLD DEPOSITS

During the week of September 3, 1932, the writer made a second examination of the Red Flats Gold Deposits, more particularly to re-sample certain sections of the property, also to check the salient features of his report made upon this property under date of September 27, 1932. The results of the re-sampling were as follows-

Ainlay Gold Centrifugal Separator Tests
Gold per ton of 2000 pounds

No. 1.	Composite sample over a large section of Northeasterly exposures. 9 pounds treated.	Gold $1.36 per ton.
No. 1-A	1000 feet southerly from No. 1 8 pounds treated.	Gold $2.75 per ton.
No. 2.	1000 feet southerly from No. 1 3 pounds treated.	Gold $2.00 per ton.

No. 3.	1000 feet southerly from No. 2 5 pounds treated.	Gold $1.20 per ton.
No. 4.	Composite sample taken at intervals of 300 feet over 1½ miles in width northerly and southerly across the surface areas of the Red Flats. 12 pounds treated.	Gold $1.50 per ton.
No. 5.	Composite sample, 3 pounds rock treated. Exposures over mile in width.	Gold $0.40 per ton.
No. 6.	California quartz, 4 pounds.	Gold $6.00 per ton.
No. 7.	Composite sample from 52 drill holes.	Gold $4.80 per ton.

The above tabulation of values gives the results of the Red Flats ores subjected to treatment by the Ainlay Centrifugal Gold Separator made in the Brooks Laboratory in Grants Pass, Oregon, recommended to be included in the flow sheet of the mill installed at this mine.

The ores of the Red Flats deposits appear to submit favorably to the operation of the Ainlay Centrifugal closed bowl. The tests made in this manner would be termed mill tests with the actual gold recovered from the pounds of ore treated instead of making the tests by fire assay. The ore was ground to a fineness of approximately 150 mesh. This was done in order to recover practically 95 percent of the gold by the use of the separators. Primary grinding of the ores, however, will be greatly reduced as shown by the screen tests as shown in this report, which greatly augment mill consumption of these ores, as well as reducing the cost of all phases of grinding. It is believed that Ball Mill installation would be the proper type of mill to employ for the grinding of these ores and our recommendations would include Ball Mill installation with the installation of a Dorr Classifier which would return the ore to the Ball Mill until the proper mesh was obtained. The flow sheet would then include the pulp discharge to a Dorr Classifier, to Ainlay bows, to concentrating rables, to amalgamation plates, to dump. If tests of the tailings reveal sufficient values cyandization to recover the values would be recommended, though it is believed a 95 percent plus recovery can be made from these ores obviating the necessity of treating the tailings.

In testing the ores of the Red Flats by the use of the Ainlay bowl, ores ground to different mesh were used. Ores ground to a 60 mesh showed a recovery of $1.95 per ton. The same ore ground to a 150 mesh (plus) gave a recovery of $2.40 per ton, showing the necessity of fine grinding.

On Page 13 of this report estimates for mill installation are given. These estimates include a Rod Mill instead of a Ball Mill installation, which may change these figures somewhat. It will be noted that a Dorr Classifier is not included in these estimates, as well as the Ainlay Bowls. We would, therefore, add an additional $15,000.00 to our estimates as outlined therein, since the Ball Mill installation will be of necessity somewhat higher in cost over that of Rod Mills, together with the Classifier and Ainlay Bowls. We did not, however, change the cost of the power units in the estimate since we believe same are amply provided for.

In summarizing the report small space has been given to the magnitude or potentialities of the Red Flats Property, aside from the tabulation made on Page 14 as to the result of the mining and milling the precious metals contained in these ores. It must be remembered that the perimeters of this property may be safely extended to cover at least 800 acres and that reference is made in the appraisement of the values of these ores only to a depth of eight feet over a small area and seven feet over the larger areas. It is the belief of the writer that the ores contained in these deposits would extend to considerable depth, at least a large part of these deposits should extend to fifty feet or more. We refer now to the decomposed iron ores. Therefore, the potentialities of this property, aside from the positive ore as stated in this report, should be very great.

A note worthy fact is that the recoveries in gold are almost double made from the drill hole tests, indicating that even at a slight depth the value in gold increases over that of the surface tests.

Insufficient reference has been given in the body of this report to what is termed the California Quartz. The character of this ore and it's favorable geology would indicate depth. The ledge, safely twenty feet in width, is traceable upon the surface as it extends northerly and southerly over a distance of two to three thousand feet. Fire assays of this quartz show a value of $1.90 to $4.60 per ton in gold. When ground to 150 mesh and treated by the Ainlay Bowl the recovery was $6.00 per ton, partly free gold as shown when panned. Four pounds of the ore were treated, constituting a composite sample across the twenty foot face of the quartz deposit. It is believed that this deposit may produce three to four dollars per ton in gold, including the usual waste dilution when milled, and in some instances running much higher values. It's width, structure, value, and magnitude as to tonnage indicates this deposit particularly to be worthy of extensive development, since it is believed that there would be little difficulty in mining and milling this ore for a total cost of not to exceed $2.50 per ton.

As to the platinum content shown by Black & Deason, Chemists of Salt Lake City, to the amount of 90¢ per ton contained in this ore, we regard as exceptional. If the platinum value is maintained at depth we would reduce platinum in this section of the country, which has produced the larger amounts of platinum on the Pacific Coast, we believe, for many years.

Telurium is also suspected in these ores and a careful analysis affecting this determination should be made.

Screen Test on Crude Ore from Red Flats.

53.5% retained on a 40 mesh screen
6% retained on a 60 mesh screen
5.5% retained on a 80 mesh screen
1% retained on a 100 mesh screen
34% passed a 100 mesh screen

Our final conclusions are that this property, recognized as a great low grade gold deposit, is worthy of immediate intensive development.

Respectfully,

WILLIAM F. HAYDEN

COPY

of

RECONNAISSANCE REPORT

RED FLATS QUICKSILVER AND GOLD MINES

GOLD BEACH DISTRICT

CURRY COUNTY

OREGON

By:

John M. Nicol
Consulting Engineer

San Francisco
California

May
1 9 4 3

LOCATION:

The area under consideration is situated approximately eight miles in an airline about southeast from Gold Beach. But by the present road the distance is over thirty miles.

The mines are on a series of flats on saddle-back ridges which form the main divide between the valleys of Hunters Creek and the Pistol River.

Altitude at the Mill Camp is 2,125 feet elevation (Aneroid). The altitude varies by a few hundred feet on the different claims and then drops off abruptly into deep valleys both east and west.

The claims are in T. 37 S. and Ranges 13 and 14 W. and are all contiguous, and are located to conform to section lines (See Sketch Map).

CLAIMS AND TITLE:

The claims have been located, and recorded, in nine company-groups of eight claims each, or 160 acres per group, making a total of 1,440 acres. Of this, 80 acres has been leased leaving 1,360 acres which is under consideration in this report.

The claims have been held for approximately nine years, and ample assessment work has been done and recorded so the titles are apparently in good order.

CLIMATE:

The coast climate of this part of Oregon is mild, but with a heavy rainfall of about 80" yearly. The winters are chilly but generally little frost or snow. There are, however, some winters that are severe, resulting in heavy falls of snow at the camp; but this is exceptional, and generally we may count on all the year work. Winters are from November to March.

TRANSPORTATION-MEDICAL FACILITIES:

There is a good natural surfaced mountain road from the mines to the coast - distance 14 miles to Main Coast State Highway; at Junction at mouth of Pistol River, road will be kept open by the Forest Service. Daily stage north and south. The nearest railroad is at Coquille, 70 miles to the north or at Eureka 180 miles south.

Gold Beach, the county seat, has postal and telephone service. There are good stores, garage, repair shops, gasoline and oil supplies, hotel and auto camps, etc. There is one doctor and a small hospital.

Marshfield, 100 miles to the north is the nearest city with railway, hospital, machine shops and large stores.

Lumber can be obtained not far from camp at $30M. All other supplies must come from Gold Beach or other points. In normal times there are also coasting schooners touching at Crescent City and Port Orford - but stage, trucks and autos are the standby.

POWER-WATER POWER:

There are no public service power lines in Curry County, and power for all preliminary operations will have to be by gasoline or diesel engines.

Gasoline will cost approximately 25¢ per gallon and diesel oil 9¢ per gallon, delivered at the property.

The water power situation has possibilities. Both Hunters Creek and Pistol River are large mountain streams, with relatively large flow due to the high rain fall and forest-clad mountains, and they both have considerable fall in their upper sections, so that with a small crib diversion dam and ditch lines a fall of 300 feet or 400 feet could probably be obtained, and I judge a power site for a small plant of several hundred horse power might be obtained, with a local transmission line of only a few miles. This, however, will have to be a matter of future investigation and survey.

CAMP, MILL SITES, TAILINGS, WATER SUPPLY:

There are a few old temporary buildings but no camp of any use or value. A camp would have to be built. There are several excellent camp sites, and mill sites, and there is abundance of good running water all the year, ample for 100-ton mill and all camp requirements.

There are ample sites on down grade for tailings impounding, and there are no restrictions on tailings discharge, as there is nothing on the Pistol River to the Sea to cause trouble.

GENERAL GEOLOGY:

Southern Oregon and part of Northern California were, during Cretaceous times, a large island separated from the mainland of the continent; the old sea beaches, with abundant fossil beds, are known and recognized.

There has been emergence and submergence followed by various changes in the sea level itself. There are three well marked and relatively recent sea beaches that can be traced for long sections of the coast, and quite recently there has been a slight invasion by the sea level of the river mouths, causing heavy sand and gravel bars to accumulate from the coast literal drift.

The whole of this ancient island area is made up of very old formations ranging from early Paleozoic to the Cretaceous. Part of the territory is still unsurveyed, and with the exception of some limited areas the geological survey work has been rather general, and due to the difficulties of access many details have remained unrecognized.

There are large areas of the older sedimentaries now much deformed and metamorphosed, with development of schistose slates, highly contorted and seamed with quartz. There are also areas of the more massive and thick-bedded slates with belts of sandstones, quartzites, etc. There are numerous belts of very basic intrusive rocks, Gabbro, Pyroxene, Peridotite, Dunnite, etc., with the development of complex metamorphic zones and much massive serpentine. These areas often carry the platinum group of metals, and also chrome, and some nickel and cobalt.

The whole area is noted for its great number and variety of mineral deposits, but so far there have been no really large mines developed.

The whole area has been subject to great and long continued and repeated tectonic stresses, with the development of great numbers of faults.

In the district under consideration, we find the following:

I had no opportunity to make any detail study, and the following notes are just a sketch, but I believe it will assist in a better understanding of the property.

Nearing the mouth of Pistol River, the hill to the north appears to consist mainly of bedded sedimentaries; they are said to contain fossils and one was brought to me, and one was brought to me, and tentatively I judge it to be early Cretaceous, and the whole appears to be very sharp anticline.

As we follow up the river a little above Twin Creeks, and before reaching Slade Creek, there is a marked break in the formation, and at this point there is a great mass of rock locally called a dike. On examining this, it proved to be an auto-clastic breccia, evidently formed by a major fault - probably - at the juncture of the Cretaceous to the west and the older and more massive and resistant Paleozoic to the east. It included boulders and fragments of serpentine, the basic igneous rocks and many schists, etc. From here east and northwards, the cuts on the road show partly decomposed massive blocky slates - dikes of weathered porphyry, occasional serpentine and some zones of the basic intrusives.

As the road leaves the valley and climbs to the ridge that divides Hunters Creek to the west from the branches of Pistol River to the east, there is a marked change in the formation. The soil is barren and mineralized and only supports a scanty vegetation.

On nearing the property there is a stretch of over half a mile, where the formation is a highly contorted schistose graphitic slate, seamed with quartz, and evidently part of a basement of much older rocks. There is nothing by which these rocks can definitely be identified in age, except their relative position and general appearance.

On the border line of the property the same blocky slates appear, and I suspect we are dealing with two limbs of an anticline, of which the core is the schistose area.

From this point onwards there has been a complex of basic intrusives, and only with difficulty can some remnants of roof pendants of the slates be identified and these are altered and "baked".

On the extreme northeast, and just outside the bounds of the property, is a sharp irregular shaped hill called the "Bad Lands". It evidently carries some mineral substance that is destructive of vegetable life, because not a single blade of grass or fragment of plant life of any kind grows on it.

It is evidently one of the basic intrusives but has been subject to such intensive tectonic stress that there has been developed a super-induced platey or laminated structure; and on separating these plates they are criss-crossed every 1/4 to 1/2 inch or so, with networks of minute seams of asbestos. It is a most unusual structure.

The whole hill is said to carry gold and quicksilver in economic amounts.

It is my impression that the whole area of the claims has, in relatively geologically-recent times, been subjected to a major fault movement, nearly two miles wide, with great differential stress, developing an almost complete lamination and brecciation to profound depths. Then throughout this porous mass there has ascended either by hydrothermal or pneumatoltic processes the gold and quicksilver, which is the economic objective of this report.

Nowhere within this area was I able to observe any definite wall, vein, seam, kidney or any usual mine structure.

The deposit is a disseminated impregnation deposit of rather unusual aspect.

The gold is free, and under the microscope it shows as little rods of wire gold, little triangular crystaline fragments, and little masses of entwining plates and wires and crystals. It is nearly all pin point in size, the size of a pin's head is the largest found.

The quicksilver is formed as fine floured native quicksilver. I did not find a trace of cinabar or any other compound.

My general impression is that the present surface is a blanket enrichment due to weathering and downward mechanical accumulation in the porous ground of the two metals, during a long period, and from the slow-surface erosion of the rock residue, with possible leaching and reprecipitation.

The deepest hole so far tested is 37 feet, and it is said that it was as good at the bottom as at the top. It seems doubtful if this downward enrichment will go farther than 100 feet in depth.

HISTORY:

The mine was discovered about 10 years ago by Carl Smedberg of Gold Beach. Three attempts by small make-shift plants were tried, but were not practical.

Last year independent parties took up an adjoining area, at a lower level on the slope of the North Fork of the Pistol River.

They constructed quite a camp, and a make-shift mill, which at one glance was self-evidently impracticable and was bound to be a failure; but must have taken several thousand dollars in capital.

There has never been a proper examination or a systematically mapped and recorded sampling.

APPEARANCE AND GENERAL IMPRESSION:

Most engineers would walk onto the property and walk off again, as there is nothing in sight that looks like a vein or a mine.

The interesting geologic structure and the evident sincerity of the owners and would-be operators impelled me to a more careful analysis of the situation.

That a sufficient amount of "Values" had been found to stimulate several different people to try and develop a crude attempt at mining was palpably self-evident.

That the dissemination was "very general", so that there are no means of picking, or sorting ore, or waste, was self-evident. Therefore, any idea of small cut samples must be discarded and some form of "bulk sampling" and "bulk reduction" was the only chance of really testing the value of the property.

If, on the other hand, the "owner's beliefs", tests, and statements, could be verified or discarded without too great an outlay, it would be well worth while, because of the great area involved.

A STATEMENT OF THE CASE:

Taking the workable surface area at over a thousand acres, and the depth immediately available at 10 yards, and approximately at 5000 yards to the acre, we have the great volume of 50 million cubic yards. Each yard will give more than one ton.

Apparently the mean average of various samplings - over any part of this area - is 4 lbs. quicksilver and $3.00 in gold recoverable value from bulk samples.

If an independent sampling is to be carried out, it is evident from the area involved that no "salting" would be possible, providing the investigating staff and crew took charge, and cut and reduced their own samples.

But I quite agree with the owners that small cut samples are valueless. I think the reason is that the enrichments occur in minute fracture planes and therefore the only way to average is a bulk sample.

Probably systematic testing and mapping will show some more defined areas of impregnation and some barren areas. But no present available data gives any daylight on this problem.

VARIOUS STATEMENTS AS TO SAMPLE VALUES:

Case No. 1. Over one mile area, shovel cut grab samples were taken, and thrown up into a truck until 1-1/2 tons were obtained. This was broken down and shoveled over and re-sampled down to a 200 lb. lot.

This was taken to the plant of the Western Gold and Platinum Company in San Francisco, ground to all less than 30 mesh and run over a McCartney table and concentrated down to 10 lbs. This was then dried, and cut for assay, and a sample assayed and gave returns of $910.00 per ton of concentrate, or approximately 45¢ per lbs. of concentrate. Therefore, the 10 lbs. of concentrate had a gross content of $4.50, and this was from 200 lbs. average. This was 1/10 of a ton and the final result would show $45.00 per ton.

In my opinion this is not possible and represents some error along the line.

Case No. 2. Original owner claims 230 samples from 3 feet to 17 feet deep taken with a fish tail auger. Assayed results showed average on east slope of ridge of $8.00 per ton and on the west slope of $10.00 per ton.

Case No. 3. Mr. Lon Shannon, working for and with the Pacific Minerals Association, claimed that they cut several hundred random samples on many different parts of the property, using 5 lbs. samples in pot retort. Quicksilver was actually distilled off and weighed for each sample so tested. Some were low, some higher, but no blanks. Average was 1 gram recovered per lbs. of ore - - 4.6 lbs. per ton.

There has been nothing definite, systematically done or properly recorded, but I saw enough to be satisfied that gold and quicksilver occur disseminated in the fracture planes of the rock over a wide area.

Mr. John A. Shoff, who has been trying to find a means to test and develop this property for some time, and who has cut a number of samples, states that as far as he can judge the average will be about $3.00 in gold per ton (gold at $35.00) and about 4 lbs. of quicksilver.

FINAL PROOF NEEDED:

Test Methods Suggested. The testing should be carried out with relation to the geologic structure, and also with relation to the form of occurrence of the metal in the "ore".

(1) The ore will always be more or less surface material - in winter, all wet, decomposed rock, red earth, sandy material, with roots, trash and leaves, etc.

(2) The gold is exceedingly fine, and much of it amalgamates.

The quicksilver is apparently all fine floured native; so fine that it quickly floats and is lost if some cleanser or depressant is not added to the wash water.

I suggest, therefore, a scheme covering 62 shallow surface pits, not over 3 feet each, distributed approximately as per attached sketch.

Sample to a volume rather than to a tonnage basis. Treat it as a detrital or residual placer, as it is surface and "native metal".

Take 5 pan samples from each hole, dump into a small iron tub (Note: Galvanized buckets, tubs, etc. must not be used or mercury will be absorbed).

Wash, pick out and keep all floating leaves, trash, roots, etc.; eventually dry and retort as a test. Screen out all 1/4" material and pass to crusher to all-1/4"; return to iron tub (one test should be made with finer grinding in duplicate, say to 30 mesh) and add "depressant", stir, settle a few minutes and pour off excess water.

Using clean water, with a little "depressant" added, make a crude concentrate either using small table, rocker or pans.

Try to work to about 20 to 1 ratio. Dry concentrate in sun or very low heat. Retort to extract quicksilver. Weigh and record weight of residue in lbs., oz. Assay by regular methods for gold per ton of concentrate. Record results, and per lbs.

NOTE: As approx. 150 crowned full sized miners' pans = 1 cu. yd. of material, the multiplier for a 5 pan sample is 30.

THEREFORE: Weight of concentrate C x 30 = Expected average weight of "saveable" concentrate per cu. yd.

Weight of distilled drops of mercury M x 30 = weight recoverable by wet concentration and retorting of concentrates in "lbs" per cubic yd.

We now have "sample tests" to accord with a procedure that can be followed in a practical "Mill Flow Sheet" as suggested.

The operators can supply us with the equipment for this work - loaned - as follows:

A small engine, laboratory crusher and grinder and necessary C.S. belts, et

A small power driven oil burner, retorts, furnace, iron trays for drying, trucks, tools, pans, small concentrating table (?) etc. 62 samples can be brought down to San Francisco for assay.

The total outlay for this work can be kept down to a moderate amount; and the results would be sufficient to determine whether to reject all former work or to consider the property as worth further outlay for tests and possibly a small pilot mill.

SINGLE TEST CARRIED OUT:

I was alone and had not the time, strength or equipment for proper tests, so only two samples were cut as above, with the operator's assistance. Each sample of 5 pans - concentrated down by hand panning - and concentrates sent down to San Francisco for retorting and assay. Results are attached to this report.

AN ATTEMPTED PERSPECTIVE VALUATION:

To evaluate this property, the following data must be obtained:

Proper sampling as above suggested, with a map plotted to show zone values and some formula developed to show a real average valuation.

The addition of at least three shafts to say 50 feet each, sunk with windlass, and sampled as above every 10 feet or 15 additional samples.

A laboratory mill test run.

Given the results approximate the owners and operators claims - we then have the possibility of "Plus" 50 million cubic yds. of surface material. Then with a 100 ton a day mill we would have a possibility of; at normal extraction values and at present prices of;

400 lbs. Quicksilver @ $2.00 per lb.	$800.00
Gold @ $35.00 oz.	300.00
Daily gross mill return	$1,100.00

or a probable gross of $20,000 to $30,000 per month.

A TENTATIVE ORE TREATMENT PLAN AND MILL FLOW SHEET.

It may seem premature to suggest designs for a mill while the value of the mine, as a mine, is still undetermined, yet due to the war and the price and urgent need of quicksilver, the mine should be opened up with a minimum of delay.

As this type of deposit is very unusual, the success or failure will in a great measure depend on a pre-determined plan of both mining and ore treatment that shows a logical sequence to the data obtained by testing.

"It is to be noted that this is the only quicksilver mine that I have seen or heard of where all the quicksilver occurs as - "floured native" - and where it is all disseminated on the surface so that it can be sampled like a placer".

With a return to normal markets after the war, quicksilver will probably drop to $60 or $80 a flask, and then the gold will be as important or possibly more important than the quicksilver.

The tests so far made seem to show that the effect of retorting, with the roasting of the ore, increased the output of gold.

Possibly there are Selenides or Tellurides.

Furthermore, all the ore will be surface material, leaves, roots, trash, etc., and for six or seven months of the year it will be very wet.

There is such a vast volume of material, that it would seem that high extraction, while always desirable, may not be as important as low capital cost for a mill having large capacity and great simplicity and certainty in treatment.

Cinabar ores do not lend themselves to any form of crushing and wet concentration or flotation, as the bulk of the soft cinabar goes into a "Slime-Paint" that will not coalesce, whereas "native" can be brought together and is coalesced by activating depressants.

Due to the surface mining and the mud and roots, the ore as brought to the mill by trucks cannot be handled by any of the usual bins or ore feeders.

The indications are that the following will be about the general outlines to be adopted:

Some form of surface mining trucking to a mill site where water and dump is available for tailings. Truck-dumping onto a sloping receiving platform.

A surface drag feeder, or hand feeding by shoveling to a large scrubber over an "8" bar grizzley. All "8" to be sledge broken.

Scrubber to have a capacity of at least 10 tons per hour per unit, and must scrub submerged.

Discharge of scrubber to heavy wire trommel screen to 1/4" mesh.

All oversize to crushers, - fineness by test - crushed product returned to circuit. Storage bins can be inverted between stages of crushing with standard feeders and equipment, as here the ore will be clean and dewatered rock.

All screened undersized and all water from trommel to go to the

"DRUM ROUGHING CONCENTRATORS"- These are suggested for this work, as against tables or jigs for the following reasons:

Part of the "values" to be saved are very fine, but there are also heavy grains of concentrates up to 1/4"; bulk concentration of an unsized product, with a ratio possibly as high as 20 to 1, or even higher, is therefore called for.

The principle of concentration in the "Drum" is one where deep riffles are used and the movement causing settlement is, and can be adjusted, entirely independently of the motion causing removal of the concentrates. This latter is due to the rotation of the drum, and the longitudinal pockets, which are rotated out of the zone of concentration as fast as the concentrates settle.

This type of concentrator does not make a separation, or a clean concentrate; it is strictly a bulk, volume rougher, for an unsized product. It has a very wide range of adjustment, but requires little attention.

The concentrates would have to be dried at a low temperature to avoid loss of quicksilver. The dried product would then be treated by distillation in "D" retorts.

A bank of 10 D retorts should be able to treat all the concentrates from 120 to 150 tons a day.

But we will base the first economic estimate at 100 tons a day mill capacity.

The dried concentrates can be treated in a small simple cyanide plant, to extract the gold.

Probably a small sand percolation plant will be about all that will be required.

The following further suggestions are offered, but they will be a matter of test and experiment.

Between the discharge of the scrubber and the intake of the trommel screen, it would be a distinct advantage if some means could be obtained of separating roots from the rocks. It is possible that this may not be as important as it at first appears. But it is necessary to watch for it.

The feeding of a small amount of activating depressant could be arranged between the screen and the concentrators.

The failings from the concentrators could be passed into a simple inclined dewaterer with a warm screw in the bottom. I have built and used these before and they are not costly and simple to operate.

The tailings are delivered in a dewatered condition from beyond a baffle. The greater part of the water can be drawn off by an automatically adjustable side spigot so that only a small volume of the surface water passes over an edge wire. This water will contain any float quicksilver or gold and can be treated apart with depressants in a small vat, and the residue filtered, dried and sent to retorts.

In the dewaterer, there is a good settling point for any chance gold, amalgam or quicksilver that has escaped the concentrators.

The above is a suggested general outline; it is practical and will have a large capacity for a relatively small capital outlay.

SUMMARY AND CONCLUSIONS

An unusual deposit, of great extent, with no properly recorded data as to its merit or value. Required careful testing that will be above question but not by small cut samples. Until this has been done no condemnation or appraisal is permissible. It should not be turned down offhand because it is unusual. I feel there is enough favorable evidence to warrant the above suggested sampling.

Submitted;
/s/ John M. Nicol
Consulting Engineer

Process for Recovery of Nickel, Cobalt and Copper from Domestic Laterites

by RICHARD E. SIEMENS
Project Leader—Nickel Research, Albany Metallurgy Research Center

and JOHN D. CORRICK
Physical Scientist, Division of Ferrous Metals, U.S. Bureau of Mines

The United States produced less than 10 percent of the nickel it consumed during 1975. The importation of nickel required to make up the remaining 90 percent cost the nation $670 million. U.S. reserves of nickel amount to less than 0.5 percent of the world's total reserves, however, U.S. resources account for over 12 percent of the world's total resources.

The demand for nickel in the United States, as projected by the Bureau of Mines, will grow at an annual compounded rate of nearly 3 percent.[1] This growth will result in a demand for nickel in the year 2000 of 385,000 short tons. The only possible way for the United States to maintain its present ratio of domestic production to total consumption, or hopefully to reduce the deficit between the two, is to develop economical methods that will permit the extraction and recovery of nickel from domestic resources. This, then, is the objective of current research being conducted at the Albany Metallurgy Research Center on domestic laterites.

U.S. nickel resources estimated at 15 million tons

U.S. resources of nickel are estimated by the U.S. Geological Survey at over 15 million short tons of contained nickel.[2] The largest single resource is that found in the low-grade Duluth gabbro of northeastern Minnesota near the town of Ely. The nickel silicate and laterite deposits found in southern Oregon and northern California account for the second largest nickel resource in the United States. In fact, this area supports the only domestic nickel mine and smelter now in operation. Other areas that contain significant quantities of nickel are Alaska (principally under Brady Glacier in Glacier Bay National Monument), Montana (Stillwater Complex) and Washington (Cle Elum District).

The nickel silicate deposit near Riddle, Douglas County, Or., was discovered in 1846. The Bureau of Mines currently recognizes this deposit as containing the only mated to contain about 200,000 short tons of nickel. The total resource of the district is considerably larger but with a lower nickel content.

There are additional lateritic resources in Oregon and California of the nickeliferous, iron-laterite type similar to that which predominates in Cuba and the Philippines. The principal lateritic deposit in Oregon is located on Eight Dollar Mountain. Other significant lateritic resources have been identified in the Woodcock Mountain area of Josephine County and the Red Flats area, east of Gold Beach, Curry County, Or.

Deposits of low-grade nickeliferous iron laterites are known in northwestern California near the Oregon border. Pine Flat Mountain northeast of Crescent City, Ca., is probably the best known district. The largest resource area in California is in the Little Red Mountain district in Mendocino County.

Richard E. Siemens is project leader of the Nickel Recovery from Western Resources project, initiated in 19[illegible] [illegible] the Bureau of Mines' Albany, Or., Metallurgy Research Center. Siemens received a BS degree from Oregon State University in 1960 and joined the Bureau of Mines after several years of graduate school.

John D. Corrick is a physical scientist with the Division of Ferrous Metals, U.S. Bureau of Mines. Corrick began his bureau career in 1956 at the College Park (Md) Research Center. He has served as commodity specialist on cobalt and on nickel; state specialist on Oregon; country specialist on Spain, Greece and New Caledonia; and acting group leader for the ferroalloys branch.

Resource data underestimates extent of deposits

The most recent published data on resources of nickel in Oregon and California show nearly 565,000 short tons of nickel.[2] The data on these resources was obtained from cursory reconnaissance of the districts. Much of the sampling was done by hand augers to depths of 10 to 12 ft and supplemented with a few bulldozer trenches. Auger sampling and churn drilling are believed to give poor results because of the attendant dilution of the ore with surface soil.

Recent exploratory pit sampling by private firms of the nickeliferous iron laterites in Oregon has indicated more extensive deposits of nickel than were thought to exist previously. Visual observations of the areas being sampled and discussions with operators working there indicate substantial quantities of nickel laterite ore. Pit sampling has shown the nickel bearing laterites to exist at depths four to five times greater than was assumed in earlier evaluations of the area. Nevertheless, much of the exploratory work is in the formative stages and a final quantitative evaluation of the area cannot be made until the exploration is complete and the reports assessed.

The domestic laterites contain from 0.5 to 1.2 percent nickel, primarily in the mineral goethite, and 0.06 to 0.25 percent cobalt in a manganese oxide wad. About 2 percent chromium is present as chromite. Other metal contents in weight-percent are: Fe, 36.1; Mn, 0.5; Zn, 0.04; MgO, 7.2; SiO_2, 21.5; Al_2O_3, 4.11.

Treatment processes on domestic laterite uneconomic

Commercial processes used to treat laterites[3] are generally unacceptable for economic and efficient extraction of nickel and cobalt from domestic laterites. Pyrometallurgical processes were ruled out because of the high electrical energy requirements and because the chemical composition of the laterites was not suitable for economical application of these approaches. Acid leaching procedures would not be economical because of the relatively high magnesia content of domestic laterites. The general approach of reduction leaching has a wider range of application to lateritic materials. Improvement of existing reduction leaching processes appeared to have the greatest potential for processing domestic laterites.

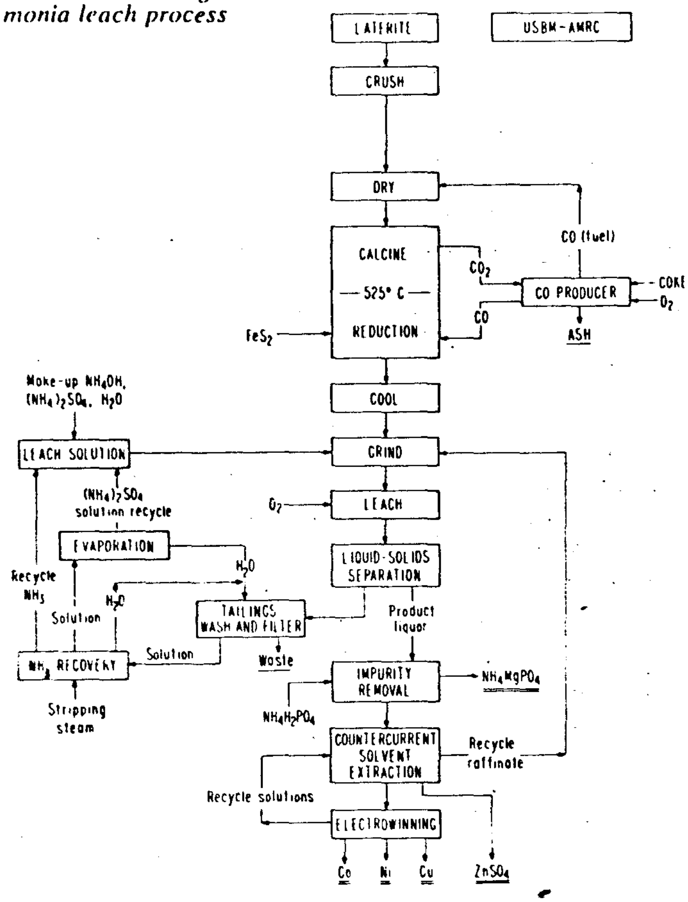
The major disadvantages of the procedure at Nicaro, Cuba,[3,4] are:

1) Relatively high reduction temperature (700–760°C) and long reduction retention time of 90 minutes
2) Form of product (nickel oxide which is not as desirable as metal for many applications)
3) Contamination of the nickel product with cobalt, and to a lesser extent with impurities such as zinc, manganese, magnesium and copper
4) Low recovery, about 73 percent Ni.

Efforts were initiated in 1957[5] to remove the cobalt as a mixed nickel-cobalt sulfide. This modification and others have been incorporated into new operations—Greenvale, Australia, and Marinduque's Philippine operation.[6,7]

Significant improvements have resulted from the modifications, such as the cobalt extraction as a mixed nickel-cobalt sulfide and an upgrading of the primary product. In regard to the latter, the nickel content of the oxide product is increased from about 72 to about 90 percent at Greenvale, while high-temperature, high-pressure hydrogen reduction is utilized by Marinduque to recover pure nickel.

Fig. 1. USBM selective reduction-oxidizing ammonia leach process

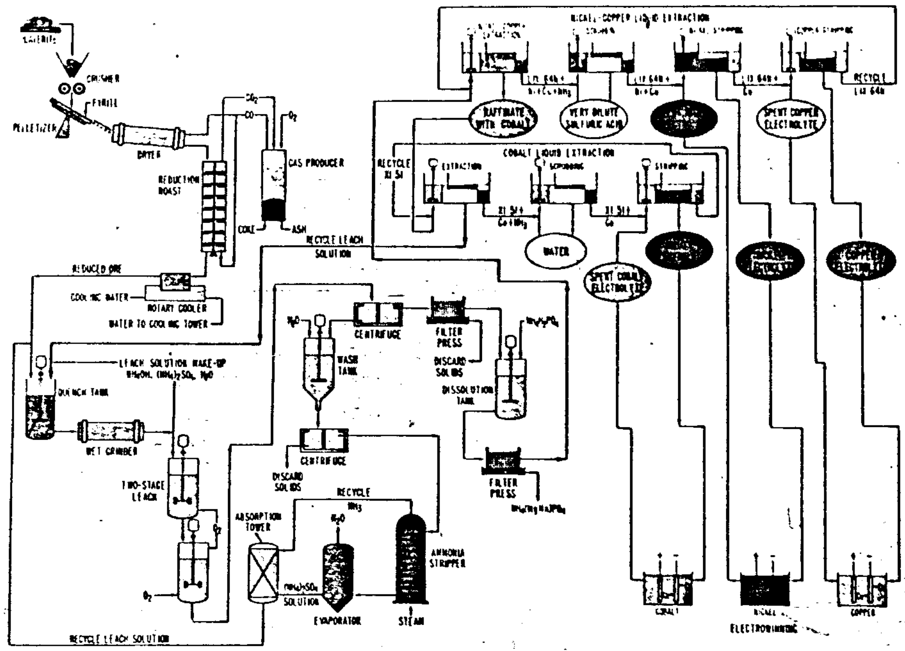


The disadvantages of the newer approaches are that the high temperature reduction conditions, as used at Nicaro, are essentially unchanged, and the net recovery is still very low because 9 to 15 percent of the extracted nickel is recovered with the cobalt. The cobalt recovery is low (60 to 65 percent) and cobalt is not recovered in a usable form. The mixed nickel-cobalt sulfide must be further processed to separate and recover the nickel and cobalt.

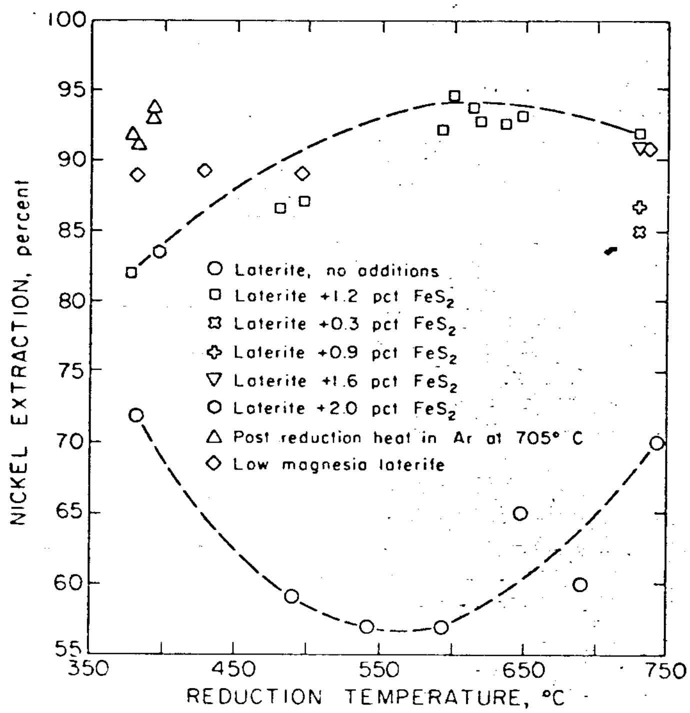
Recent modifications by Universal Oil Products Co. have resulted in improved extraction, but the improved extraction is still subject to reduced recovery by 9 to 15 percent in the cobalt extraction procedure discussed above.[8-11]

USBM recovery process incorporates selective reduction

A process under development by the U.S. Bureau of Mines to recover nickel from domestic laterites also incorporates reduction roasting and leaching but has significant advantages over the Nicaro procedure and modifications thereof. The process incorporates selective reduction and an oxidizing ammonia-ammonium sulfate leach with solvent extraction and electrowinning to recover nickel in cathode form (fig. 1). Cobalt and copper are also recovered in cathode form and zinc, if present, is recovered as zinc sulfate. To avoid their build-up in recycle, low



Nickel-copper-cobalt recovery from domestic laterite



Effect of pyrite additions on nickel extraction

concentrations of magnesium and manganese in solution are removed as a marketable ammonium magnesium-manganese phosphate fertilizer. A semi-continuous processing circuit capable of treating 20 dry lb of laterite per hour is in operation at the Bureau's Albany, Or., Metallurgy Research Center.[12, 13]

Nickel extraction higher with pyrite additions

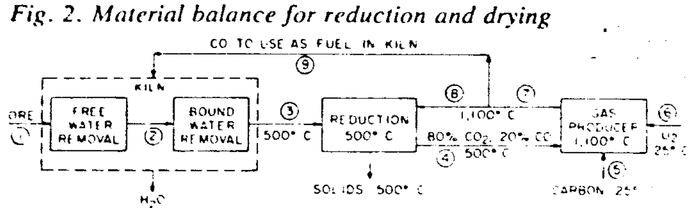
The laterite is crushed to minus 1/4 in., dried to remove both free water (15–40 percent) and water chemically bound to the iron oxide (about 13.5 percent), and blended with about 1.2 percent pyrite. The magnesia content of the domestic laterites is relatively high (about seven percent) and becomes involved in a secondary reaction with reduced nickel to form inert nickel magnesium silicates. Sulfur inhibits this reaction and is conveniently added to the laterite as pyrite. Nickel extraction from domestic laterites was shown to be 50 to 60 percent higher with 1.2 percent [12]pyrite additions for reduction temperatures from 380 to 525°C. Sulfur forms are readily absorbed by laterite and report to the tailings.

Along with the pyrite additions, the significant variation in the reduction procedure adopted by the Bureau of Mines is the use of pure carbon monoxide for reduction rather than producer gas. The advantages of using pure carbon monoxide are:

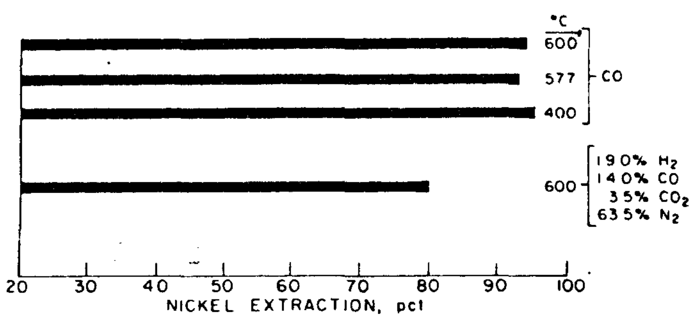
1) Effective reduction can be accomplished at a relatively low reduction temperature, 525°C, which is 200 to 300°C lower than reduction temperatures used with Nicaro type processes (at this reduction temperature, about 10 percent more nickel and cobalt were extracted using pure carbon monoxide rather than producer gas)
2) The reduction retention time is only 15 minutes, as opposed to about 90 minutes for the Nicaro process
3) The offgas is carbon dioxide, which can be recycled to regenerate carbon monoxide in a gas producer. Excess carbon monoxide can be used as fuel for drying.

A material balance for this section of the process is presented in fig. 2. As fig. 2 shows, 146.56 lb of carbon mon-

Fig. 2. Material balance for reduction and drying



STREAM	AMOUNT	IDENTIFICATION
1	2,532 LB	NEW ORE
2	2,000 LB	ORE WITH FREE WATER REMOVED
3	1,730 LB	ORE WITH ALL WATER REMOVED
4	2303 LB CO_2 366 LB CO	OFF GAS FROM REDUCTION 80 mole-pct CO_2 20 mole-pct CO
5	2466 LB C	CARBON TO GAS PRODUCER
6	2450 LB O_2	O_2 TO GAS PRODUCER
7	7585 LB CO	CO FROM GAS PRODUCER
8	[illegible]83[illegible] LB CO	CO TO REDUCTION
9	5753 LB CO	CO TO KILN TO BE USED AS FUEL



Comparison of CO and producer gas as reductants

oxide is required for reduction per dry ton of laterite. This represents an energy consumption of about 90 million Btu per ton of nickel recovered for 90 percent recovery from laterite containing only 1.0 percent nickel.

Controlled oxidation process employs two leach stages

Coupled with selective reduction, high extraction of nickel and cobalt is realized by controlling the oxidation during leaching. The leach pulp contains about 15 percent solids, 100 gpl NH_4OH and 300 gpl $(NH_4)_2SO_4$. In the controlled oxidation procedure, two leach stages are employed. In the first stage, the oxidation rate is controlled by monitoring the oxygen offgas so that reduced nickel, cobalt and copper are completely solubilized as ammonia complexes while iron is solubilized in the ferrous oxidation state. This procedure is important to achieve high extraction since most of the reduced nickel is present as ferronickel particles. Without controlled oxidation, iron oxide films form on the particles and impede further solubilization.

In the second stage, iron is further oxidized to form insoluble ferric hydroxide. Laboratory tests have shown that control of the oxidation rate also decreases the possibility of nickel, cobalt and copper being occluded in the iron hydroxide precipitate.

The following simplified equations illustrate the chemical reactions involved:

Stage 1 $Ni^\circ + 1/2\,O_2 + 4NH_3 + H_2O + (NH_4)_2SO_4 = [Ni(NH_3)_6]^{++} + SO_4^{--} + 2\,H_2O$

$Fe^\circ + 1/2\,O_2 + H_2O \rightarrow Fe^{++} + 2(OH)^-$

Stage 2 $4\,Fe^{--} + O_2 + 2H_2O + SOH^- = 4\,Fe(OH)_3$

In stage one, the formation of nickel hexammine is illustrated. In general, for nickel, the amine formed is $Ni(NH_3)_x^{++}$ where x varies from 2 to 6. Similar reactions occur for cobalt and copper. In a continuous processing circuit, the above reduction-leach procedure applied to Pine Flat laterite results in a pregnant leach solution containing in gpl: Ni (2.2), Co (0.82), Cu (0.1), Zn (0.08), Mg (0.3) and Mn (0.1).

Centrifuge cakes require less washing

Although several liquid-solid separation procedures could be applied to the laterite leach solutions, the Bureau of Mines is investigating the feasibility of using centrifuges. In the continuous processing circuit operated by the bureau and in tests conducted by a commercial centrifuge supplier, 70 to 80 percent solids were obtained in the centrifuge discharge without flocculation. These centrifuge cakes require considerably less washing than filter cake or thickener underflows that contain 40 to 50 percent solids.

In bureau tests, only about 4 percent of the metals extracted and 4 to 8 percent of the NH_4OH and $(NH_4)_2SO_4$ present in the leach solution must be washed from the centrifuge cakes. Washing on a tilting-pan filter, even without repulping, resulted in a discharge cake containing only 0.04 percent each of the extracted nickel and cobalt and about 0.13 percent each of the leach reagents.

To recycle the wash solution, NH_3 is first removed by steam stripping, leaving a dilute ammonium sulfate solution. The water added in washing is then evaporated, and recovered NH_3 is recombined with the proper strength $(NH_4)_2SO_4$ solution in an absorption tower.

Overflow solids in the pregnant centrate (about 1.5 percent) are removed with a plate and frame filter. The polished solution is then contacted with an ammonium phosphate fertilizer product to precipitate the small concentrations of magnesium and manganese. This precipitate is removed with a plate and frame filter and is a marketable fertilizer product.

Nickel extracted in three stages from leach solution

A countercurrent system of mixers and settlers is used to separate and concentrate the metals in solution. Nickel and any copper present are extracted from the leach solution in three stages with a 12-volume-percent solution of LIX-64N in kerosine (fig. 3). Since cobalt is present in trivalent form after the oxidizing leach, it is not extracted. Zinc is crowded from the loaded organic in one stage and entrained ammonia is washed from the organic in two stages with dilute (pH 3.5 to 4) sulfuric acid and $(NH_4)_2SO_4$. A bleed stream from the washing circuit is recycled as required to return $(NH_4)_2SO_4$ to the leaching circuit.

Nickel stripping, in five stages at the level of free acid concentration in the spent nickel electrolyte (about 5

	With diaphragms	Without diaphragms
Electrolyte feed	90 g/l Ni^{++}, 80 g/l Na_2SO_4, 20 g/l H_3BO_3, pH 3.5	90 g/l Ni^{++}, pH 3.5
Approximate feed rate	12 ml/min/sq ft cathode	72 ml/min/sq ft cathode
Cell conditions	catholyte pH 3.5	pH 1.9
Internal recirculation	None	1.4 l/min/sq ft cathode
Cell temperature	50°C	50°C
Electrode spacing	3.5 in.	2.0 in
Volts*	4.0	3.2
Current density	19.5 amp/sq ft	19.8 amp/sq ft
Duration of run	2160 min	320 min
Current efficiency	99.4 pct	65.5 pct
Power consumption**	1.667 kwh/lb	2.025 kwh/lb
Nickel drop, est.	29 g/pl	3.3 g/pl

*Across hanger strap and anode

**For nickel deposition only

Table 1. Electrowinning of nickel with and without diaphragms

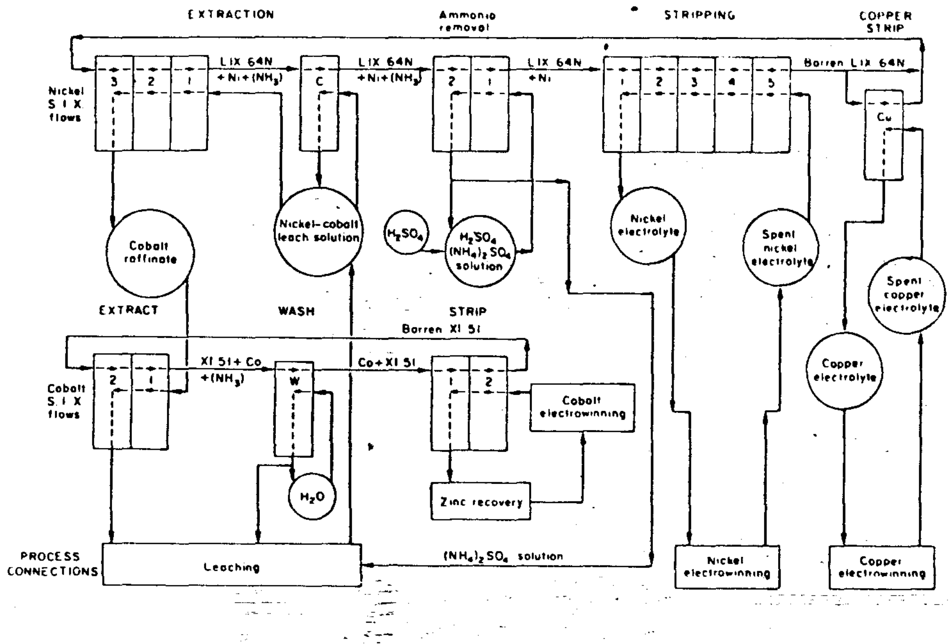


Fig. 3. Separation and recovery of nickel, cobalt and copper from ammoniacal leach solutions

gpl), is selective, leaving the copper loaded on the organic. The copper is then stripped from the organic with spent copper electrolyte containing about 175 gpl sulfuric acid. Cobalt is extracted from the nickel-copper-free raffinate in two stages with XI-51, dissolved as a 10 volume-percent mix with 10 volume-percent isodecanol and 80 volume-percent kerosine. Entrained ammonia is washed from this organic with water in a single stage. A bleed stream from this wash could be returned either to the leaching circuit or to the ammonia recovery circuit. The cobalt, along with any zinc present, is stripped in two stages from the organic with spent cobalt electrolyte. Zinc is extracted from the acid strip solution with di-(2-ethylhexyl) phosphoric acid and recovered as zinc sulfate. Trace concentrations of organic are removed from raffinate streams with polypropylene coalescers.

Efforts being made to economize non-diaphragm procedure

Nickel, cobalt and copper are all electrowon from the acid strip solutions. The concentrations required for electrowinning are provided by adjusting the organic to aqueous volume ratio in stripping. Starter sheets were prepared on titanium blanks. The conditions and results of electrowinning nickel with and without diaphragms are presented in table 1. The conditions for electrowinning without diaphragms with a 3.3 gpl nickel drop per pass through the cell are similar to those employed by SEC Corp., El Paso, Tx.[14]

Diaphragms have been traditionally used for electrowinning nickel to isolate the acid from the cathode and result in high current efficiency and high nickel drops (table 1). However, the capital cost of a diaphragm tank house is higher than for the non-diaphragm type, and diaphragm maintenance is a concern. Therefore, efforts have been directed toward making the non-diaphragm procedure more economical.

In recent research at the Bureau of Mines with non-diaphragm cells, the nickel drop has been increased to 4.2 gpl, the current efficiency increased to 84.4 percent, and the power consumption lowered to 1.4 kwhr per lb (36 percent decrease) by operating the cell at 85°C. By decreasing the cell-feed rate by one quarter, the nickel drop was increased to over 12 gpl at the same current density, at a current efficiency of 71 percent and a power consumption of 1.63 kwhr/lb. Under these conditions, the nickel drop is still not as high as for diaphragm operation, but it is nearly four times that of the SEC operation, and the power consumption is about the same as that for diaphragm cells.

In evaluating another possibility, production was nearly doubled by increasing the current density to 39 amp per sq ft and doubling the last flow mentioned (still half of that in the table). As a result, the power consumption was increased to 2.03 kwhr per lb (about the same as the 50° C run in table 1 but at twice the production) and the current efficiency lowered to 69.6 percent. Tests similar to the latter one have been run for up to 27 hours without dendrite formation, even without agitation of the cell solution as done by SEC.

Buffering action increases performance

The increased performance is made possible by a buffering action at the higher temperature which can be obtained to a lesser degree by adding sodium sulfate. Thus, several options are available in the nickel electrowinning, but the nickel drop and current efficiency can definitely be increased over the SEC values at a substantial electrowinning power savings.

The increased electrolyte temperature can be achieved by heat exchangers utilizing heat from the furnacing. The low-acid concentration in the nickel electrolyte would not affect the hangers as much as high-acid copper electrolytes, but this potential disadvantage of operating at higher temperatures will have to be further evaluated. Cobalt was successfully electrowon in non-diaphragm cells under conditions similar to those for nickel. Copper will be electrowon under conditions similar to those commonly applied in solvent extraction-electrowinning plants.

The above described process has been used to extract up to 92.7 percent nickel and 91.4 percent cobalt in continuous-circuit operation from domestic laterite containing only 0.73 percent nickel, 0.2 percent cobalt. As a comparison, 95 percent nickel and 70 percent cobalt was

Material	Content, percent	Reduction temp., °C	Extraction, percent			
			Ni	Co	Cu	Zn
Domestic laterite	Ni (1.0), Co (0.2), Cu (.05), MgO (7.2), Fe (36)	525	92	87	82	
Domestic laterite	Ni (0.73), Co (0.2), MgO (6.5), Fe (36)	525	92.7*	91.4*		
Domestic laterite	Ni (0.53), Co (0.06), MgO (5), Fe (30)	650	85.2	76		
Domestic Serpentine	Ni (0.4), Co (0.03), MgO (16.0), Fe (16), SiO_2 (33)	700	78	52		
Philippine laterite	Ni (1.15), Co (0.13), MgO (1.25), Fe (47)	400	95	70		
Stage, oxidation-roasted Stillwater concentrate	Ni (1.26), Cu (1.38) (Conc. anal.)	770	94		96	
Spent nylon catalyst	Ni (8), Zn (27)	Remove organic with solvent, no reduction	100			100
Stage oxidation-roasted tailings concentrate	Ni (0.16), Cu (2.86), Co (2.1) (conc. anal.)	700	86	87	92	
Dead roasted chalcopyrite	Cu (36)	350			98	
Oxide copper ore	Cu (5.6), Fe (18)	520			85	
Spent margarine hydrogenation catalyst	Ni (22) (organic burned off)	600	95			
Rayon catalyst	Zn (34)	450				92
Brass mill process dust	Zn (10.9), Cu (5.2), Ni (0.07)	395	57		95	97.5
Mn sea nodules	Ni (1.4), Co (0.2), Cu (1.1)	380	90.2	76	94.3	
Granulated crude ferronickel	Ni (12), Co (2.5), Fe (83)	No reduction	97.7	89.6		

*With current refinements in semi-continuous process development unit

Table 2. Application of USBM reduction-leach, solvent extraction-electrowinning process

extracted from Philippine laterite with the process, with or without pyrite additions, after reduction at only 400° C. The Philippine laterite contained 1.15 percent nickel, 0.13 percent cobalt and only 1.25 percent magnesia.

Although the process was developed primarily for domestic laterites, it has also been used to extract high percentages of Ni, Co, Cu and Zn from a variety of oxide and dead-roasted sulfide materials. The extraction and reduction temperature used for some of these materials is summarized in table 2.

USBM process achieves high metal recovery

The Bureau of Mines' selective reduction-oxidizing ammonia leach process has successfully met the processing goals sought:

1) High metal recovery (about 90 percent)
2) Relatively low energy requirements (about 90 million Btu per ton of nickel recovered from laterite containing only one percent nickel for reduction and 9.6 to 13.6 million Btu per ton of nickel for nickel electrowinning)
3. Selective to the metals of interest
4) No polluting discharges
5) Reagents are efficiently utilized (reduction offgas and all leach and solvent extraction reagents are recycled)
6) The major products produced in high purity (over 99.9 percent metal cathodes) and thus readily marketable form
7) The process is efficiently applicable to a variety of Ni, Co, Cu and Zn bearing oxide and sulfide ores and secondary sources.

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Production

The final product of the smelter is ferronickel containing 50 percent Ni. Production records of the operation are listed in Table 4.

Red Flat and Vicinity Laterite Deposits (8, Plate 1)

Location

From U.S. 101 near Gold Beach, Red Flat (Figures 5 and 6) is reached via the Hunter Creek road and the connecting prospect road in secs. 18 and 19.

History and development

Mining claims were reportedly first located in the area in 1939 by Harry Hefferley (written communication, S. J. Colebank, March 14, 1978). J. E. Morrison examined the area in 1937, reporting a few shallow trenches and a 32-ft shaft (Oregon Department of Geology and Mineral Industries, 1940). Libbey and others (1947) did reconnaissance mapping and hand-auger sampling in 1946 and 1947. U.S. Bureau of Mines personnel explored the area by preliminary hand-auger sampling in 1945 and followup star-churn drilling in 1952 and 1953. Hundhausen and others (1954) reported the results. The U.S. Bureau of Mines drilled 22 churn-drill holes 6 in. in diameter, averaging 35 ft deep but ranging from 20 to 117 ft in depth. The claims have been explored by about 5 mi of prospect roads and about 15,000 ft of bulldozer trenching. The claims are currently (1977) held by Hanna Mining Company, Red Flats Nickel Corporation, and Big Basin Nickel Corporation of Gold Beach. Appling (1955) reports sampling and mapping in the area to the north in secs. 13, 18, and 19.

A small bog near the center of SE$\frac{1}{4}$ sec. 13 contains an interesting floral assemblage typical of swampy areas in ultramafic rocks. In 1971, withdrawal from mineral entry of the entire SE$\frac{1}{4}$ sec. 13 was proposed so that it could be held as a special botanical area. Subsequent investigations delineated more completely the nickel-bearing laterites of this northern area.

Geology

The area is interpreted to be a relatively thin erosional remnant of an ultramafic thrust sheet together with a thin sheet of Colebrooke Schist overlying the Dothan-Otter Point Formation. Patches of partly serpentinized harzburgite overlie sheared serpentinite, which in turn overlies and is intermixed with thin lenses and sheets of Colebrooke Schist. This assemblage overlies relatively unaltered Dothan-Otter Point marine sediments and minor volcanic rocks. Nickel-bearing lateritic soils and saprolite have developed on the partly serpentinized harzburgite. Hotz (1964) reports that ferruginous laterites developed here from serpentinite are similar to deposits in Cuba and the Philippines. Areas of complete serpentinization and shearing, however, generally contain very little soil cover. Some areas of sheared serpentinite are reported to contain anomalous nickel (Hundhausen and others, 1954, p. 7). The ultramafic rocks have been intruded by a number of small diabase dikes, as evidenced by patches of diabase surface rubble in the soil. Slumping and landsliding are very important features in the northern, western, and southeastern extensions of lateritic soil cover.

Deposits

Hundhausen and others (1954) report the maximum depth of soil and saprolite development with or without nickel-enriched serpentinite is about 50 ft. The average depth of drilling at which the ore grade drops below 0.40 percent Ni appears to be about 27 ft, but the average thickness of the nickel-rich serpentinite deposit is 12.5 ft. A few surface patches of iron-shot accumulation occur on the untransported residual soil on the ridge top in the W$\frac{1}{2}$ sec. 30. Garnierite occurs in the nickel-enriched serpentinites a few hundred feet north of the spring near the south end of the ridge in SW$\frac{1}{4}$ sec. 30 (Hundhausen and

Figure 6. Aerial photograph of Red Flat area.

Table 5. Sample assay results, Red Flat area

Sample number ****	Assay number	Type	Depth interval (ft)	Location ¼	Location Sec.	Percent Ni	Percent Co	Percent Cr	Percent Fe
1	AJG-69	Auger	0- 7	S/NE	13	0.69	0.10	----	----
2	AJG-68	Auger	0- 6	S/NE	13	0.50	----	----	----
3	5-AGG-80-84	Auger	0- 6	NW/SE	13	0.42	----	----	----
4	6-AAG-85	Auger	2- 3	N/SE	13	1.12	----	----	----
5	1-AGG-56-62	Auger	0- 7	NE/SE	13	0.74	----	----	----
6	2-AGG-63-66	Auger	0- 4	NE/SE	13	0.70	----	----	----
7	3-AGG-67-72	Auger	0- 6	SE	13	0.58	----	----	----
8	4-AGG-73-79	Auger	0- 7	SE	13	0.94	----	----	----
9	9-AAG-88	Auger	2- 4	E/SE	13	0.45	----	----	----
10	4-NR-2-15*	Channel	4-18	SE	13	0.83	----	----	----
11	7-AAG-86	Auger	2- 6	SE	13	0.96	----	----	----
12	8-AAG-87	Auger	2- 5.5	E/SE	13	0.90	----	----	----
13	3-NR-2-14*	Chip	0- 2	NW/SW	18	0.41	----	----	----
14	5-NR-2-16-20*	Auger	0-13.5	NW/SW	18	0.86	----	----	----
15	2-NR-2-13*	Channel	0- 2	SW	18	0.55	----	----	----
16	RF-4(5-15-75)***	Auger	0- 6.6	SE/SE	13	0.34	----	----	14.0
17	RF-7(5-76)***	Auger	0- 9	S/SE/SE	13	0.69	----	0.70	21.0
18	8-NR-3-1-2*	Auger	0- 5	SW	18	0.90	----	----	----
19	6-NR-2-21-22*	Auger	0- 5.5	NW/NW	19	0.78	----	----	----
20	7-NR-2-23-25*	Auger	0- 8.0	NW/NW	19	1.11	----	----	----
21	AJG-57	Auger	5- 9.5	NW	19	1.54	----	----	----
22	AKG-19	Channel	0- 2	W/NW/NW	25	0.13	----	1.00	----
23	AJG-11	Auger	0- 2.5	W/NW	25	0.51	----	----	----
23	RF-1(5-14-75)***	Auger	0- 4.2	NW/NW	25	0.18	----	----	----
24	AJG-12	Auger	0- 5	NW	25	0.49	0.08	0.39	12.5
24	RF-3(5-14-75)***	Grab	Creek cut	NE/NW	25	0.41	----	----	----
25	AJG-62	Auger	0- 6.5	NW/NE	25	0.72	0.06	----	----
26	AJG-63	Auger	0- 5.6	N/NE	25	1.06	----	----	----
27	AJG-14	Grab	Surface	W/SW/NW	25	0.75	----	----	----
27	RF-2(5-14-75)***	Grab	Surface	SW/NW	25	0.21	----	----	----
28	AJG-60-61	Auger	0- 9.5	E/NW	25	0.63	0.11	1.72	41.2
29	AJG-58-59	Auger	0- 9.5	E/NW	25	0.61	----	----	----
30	DH 6704**	Drill	0-10	NW	30	0.65	----	----	----
31	DH 6701**	Drill	0-25	SW/NW	30	0.78	----	----	----
32	DH 6705**	Drill	0-15	NW	30	0.57	----	----	----
33	DH 6702**	Drill	0-25	SE/NW	30	0.83	----	----	----
34	DH 6703**	Drill	0-15	NE/SW	30	0.66	----	----	----
35	DH 6706**	Drill	0-20	SE	30	0.55	----	----	----

* Appling (1955)
** Drill hole data furnished by Red Flats Nickel Corp; results averaged from 5-ft interval assays
*** Assayed by Hanna Mining Company
**** These numbers are found in Figure 5 and indicate locations from which samples were taken

others, 1954, p. 7). The presence of garnierite in the serpentinite probably best explains its anomalous nickel content.

The total area of lateritic soil cover shown in Figure 5 is about 1,100 acres. The unexamined area (mapped only from aerial photographs) north of Hunter Creek (largely in sec. 7) contains about 125 acres.

The average grade of soil and saprolite, based on a large number of samples over the entire area (south of Hunter Creek), is about 0.80 percent Ni, 0.15 percent Co, 1.14 percent Cr_2O_3, and 18 percent Fe. The average amount of unweathered rock in the soil over this area is estimated to be 40 percent by volume. The average depth of soil and saprolite is estimated to be about 8 ft. If more areas of nickel-enriched serpentinite are found, this figure will increase.

Three bulk samples submitted in 1975 by the Red Flats Nickel Corporation to the U.S. Bureau of Mines averaged 0.67 percent Ni, 0.06 percent Co, 28 percent Fe, 0.01 percent Cu, and 1.36 percent Cr.

Sample assay data obtained during this study and from other unpublished sources are given in Table 5. Considerably more Red Flat assay data are available in Hundhausen and others (1954).

Deposits in the Josephine Ultramafic Sheet

General

The accepted geologic name for the largest mass of peridotite and serpentinite exposed in Oregon is the "Josephine ultramafic sheet." It extends from the area of Eight Dollar Mountain, which is southwest of Selma, for about 22 mi south-southwest into California, and from Woodcock Mountain, west of Cave Junction, for about 16 mi west to the Vulcan Peak area. Approximately 180 sq mi lie within Oregon.

Fifteen areas of nickel laterite in the Josephine ultramafic sheet are described in alphabetical order in this report.

Baldface Ridge laterites (21, Plate 1)

Location: Thirteen small patches of lateritic soil are plotted on the Baldface Ridge area map (Figure 7) and occur in secs. 19, 20, 29, 30, and 31, T. 40 S., R. 10 W., and in secs. 24, 25, and 36, T. 40 S., R. 11 W., between Baldface and Chrome Creeks, both of which drain into the North Fork of Smith River. Access from O'Brien on U.S. 199 is by the Wimer road, Sourdough Chrome Mine road, and a trail extending out on the ridge from the Sourdough Chrome Mine.

The area has not been adequately field checked. Mapping was done with the aid of color infrared aerial photographs; outlines of soil areas are subject to corrections with more detailed field mapping.

Geology: The area is underlain by partly serpentinized harzburgite thrust over Late Jurassic marine sediments of the Dothan Formation to the west. The peridotite has been intruded by occasional dikes of dacitic to diabasic composition. Patches of bouldery, lateritic soil on the ridge are probably erosional remnants of a more extensive deposit on the upland surface. Patches on the lower slopes appear to be slumps or slide deposits.

Deposits: Very little specific information on the deposits is available. Those examined along the trail appear to be shallow and rocky. The total area of soil in 13 small patches determined mainly from aerial photographs is about 280 acres. Much of this may be too thin and rocky to be of commercial interest. A small patch of soil near the northeast edge of the map area on the small spur ridge at an elevation of about 2,400 ft has been recently claimed and explored in a preliminary fashion by Inspiration Development Company. The Department has had only two samples assayed from the area. Average grade of the northeastern patch is reported to be 0.67 percent Ni.

MAGNESIUM - METAL OF THE FUTURE

A revolutionary new process has been developed over the past decade in Stockton, England by the Mineral Process Licensing Corporation (MPLC).

A unique chlorination furnace is the end result developed by the MPLC of England. This new piece of technology transforms high grade magnesite ($MgCO_3$) ore into a magnesium metal. Mr. George Comnas is the Chairman of the Board for the MPLC as well as a Director of Interstrat Resources Inc. With this common director denominator investigations are currently underway to determine if experimentation on Interstrat's MgO values are warranted.

This is significant due to the foreseeable increased World consumption of Magnesium metal. As stated in "Guide to Non-Ferrous Metals", World production of Magnesium is less than than one twentieth of that of Aluminium, which automatically increases it's unit cost. It is this higher cost which has been the main reason for its comparative lack of development. However, consumption has been growing at a comparable percentage rate with that of Aluminium. Magnesium manufacturers have long predicted a future boom in magnesium production once the production grows large enough to reduce unit cost.

The determining factor for the demand of Magnesium is technological advances that will allow economical extraction and processing. Guide to Non-Ferrous Metals states "Magnesium is about twice as expensive as aluminium but, if magnesium was only one and a half times as expensive, wholesale substitution would take place in favour of the lighter metal. Because this price ratio has never occured magnesium production has been kept low. In fact only one ton of Magnesium is produced annually for every 20 tons of Aluminium. Magnesium is at present only used when weight saving is absolutely essential, but, if more of it could be produced, the unit cost of production would fall. It is quite probable that in 20 years, when fuel saving will be even more important, magnesium will be a more familiar metal than Aluminium."

The Company's Engineer, Al Bullis P.Eng., examines Interstrat's newly acquired properties.

Speculative Implications:

Interstrat Resources Inc. has initiated preliminary studies to determine whether current technology utilized in the extraction of magnesium metal (Mg) from magnesium carbonate (Mg Co_3) can be adapted to produce magnesium metal from magnesium oxide (MgO), a current by-product in the hydro-metallurgical processing of Ni-Co-Cr laterite ore.

Recently completed metallurgical studies by Hazen Research and Kaiser Engineering have indicated that pressure acid leach extraction techniques will yield 95,000 tons MgO per year or 144 pounds MgO per ton of Ni-Cal's laterite ore. Theoretically, this magnesium oxide contains 60% magnesium metal, or 86.4 pounds Mg per ton of ore. With current market prices of $0.10 (U.S.) per pound of MgO and $1.34 (U.S.) per pound of magnesium metal, the successful conversion of by-produce MgO to Mg-metal could significantly increase the gross value of Interstrat's comparable grade ore.

To illustrate this potential, the gross value per ton of laterite, assuming current metal prices and 93% recovery, is calculated according to the following conditions:

Case I: No Magnesium Oxide/Metal Value:

Ni	93% × 0.81%	× 2000 lbs/ton ×	$ 3.29/lb		$49.57
Co	93% × 0.060%	× 2000 lbs/ton ×	14.56/lb		16.25
Cr_2O_3	93% × 2.0%	× 2000 lbs/ton ×	0.03/lb		1.12
				Gross Value Per Ton:	$66.94

Case II: Magnesium oxide produced at the rate indicated by Kaiser Engineering — Hazen Research (95,000 tons MgO per year or 144 pounds MgO per ton of laterite).

Ni		$49.57
Co		16.25
Cr_2O_3		1.12
MgO	144 lbs × $0.10/lb	14.40
	Gross Value Per Ton:	$81.34

Case III: Magnesium oxide (as produced in Case II above) converted to magnesium metal, assuming a 70% recovery rate:

Ni		$49.57
Co		16.25
Cr_2O_3		1.12
Mg	70% × (144 lbs × 60%) × $1.34	81.04
	Gross Value Per Ton:	$147.98

These preliminary figures indicate that even in the event of poor recovery in the conversion process (Case III), the gross value per ton could be greatly increased. To investigate the specific technological modifications required to adapt existing Mg Co_3 treatment processes and to project construction/operating costs and ultimate recovery rates, Interstrat Resources Inc. has hired Davy McKee (Stockton) Limited, a worldwide engineering-construction organization with proven experience in the processing of magnesium ores.

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NICKEL-BEARING LATERITE, RED FLAT, CURRY COUNTY, OREGON

by

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Introduction

Peridotite and serpentine which occupy large areas in southwestern Oregon contain small amounts of nickel. Samples analyzed by the Department have ranged from trace to 0.25 percent nickel. Pecora and Hobbs (1942)[1] give analyses of peridotite (saxonite) and serpentine on Nickel Mountain, Douglas County, Oregon, which contain from 0.08 to 0.35 percent nickel.

A lateritic red soil, developed on peridotite areas, has been stripped by erosion in many places, but there are some areas which still have substantial thicknesses. Samples of this lateritic soil obtained by the Department in 1943 and 1944 indicated that in the process of weathering of the peridotite, there has been some concentration of nickel in the laterite. This is shown also by Pecora and Hobbs[2] in samples of the red soil on Nickel Mountain where a veneer of brick-red soil averages 2 or 3 feet thick and ranges in thickness from a few inches up to 9 feet. Samples of this soil contained from 0.61 percent to 1.10 percent nickel.

In the summer of 1946 the Department started a project planned to investigate the nickel content of lateritic soils on the peridotite areas of Oregon with especial attention to the possibility of secondary enrichment of nickel in the lower part of relatively thick sections of the laterite.

The first work was done in an area of Curry County known as Red Flat placers near the headwaters of Pistol River, because this locality was reported to have a section of laterite at least 32 feet thick at one place. This preliminary report is concerned mainly with the work done at Red Flat.

Four auger holes were drilled as shown on the accompanying map. Samples were taken for each foot of depth. The drilling showed that the laterite contains some hard, unweathered boulders of peridotite, and when one of these was encountered in a drill hole, no further drilling could be done with the equipment available. The deepest hole was 11 feet in depth. In addition to the drilling, a brief geological reconnaissance of the area was made.

The nickel content of the laterite appears to increase with depth, as shown by accompanying analyses, but far too little work was done to give conclusive results. Either heavier drilling equipment or, preferably, test pits or shafts will be necessary in order to sample the laterite down to the peridotite in place. All of the samples of laterite contained chromite.

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1 Pecora, William T., and Hobbs, S. Warren, Nickel deposits near Riddle, Douglas County, Oregon: U.S. Geol. Survey Bull. 931-I (1942).

2 Idem.

A careful panning test of the laterite was made, using a composite of samples obtained in two drill holes, in order to get information on distribution of the mineral content which could be effected by gravity concentration. The test indicated that most of the nickel went into the tails along with a large part of the limonite. The magnetite and a large part of the chromite were concentrated in the heavy fraction.

Chemical and spectrographic analyses were made by L. L. Hoagland and Thomas Matthews, respectively, of the Department staff.

Mr. J. E. Morrison, mining engineer formerly with the Department, visited the Red Flat property in 1937 and sampled both the laterite and peridotite to check reported gold values. His samples returned 0.02 ounces per ton in the peridotite and a trace in the laterite. Mr. Randall Brown, geologist formerly with the Department, investigated reported mercury values at both Red Flat and the Chapin property, just east of Red Flat proper, in 1942. His samples returned traces of mercury. During the war period, engineers of both the U.S. Bureau of Mines and the War Production Board examined the Red Flat area. There has been no commercial production.

Location

Red Flat is about 8 miles southeast of Gold Beach, Oregon, as shown on the accompanying map. The area lies west of the North Fork of the Pistol River and is 14 miles by way of the Pistol River road from Pistol River post office on the Coast Highway (U.S. 101). The Pistol River road is graded and drained but was in only fair condition at the time of the investigation in late May and early June. Most of Red Flat is in secs. 19 and 30, T. 37S.,R. 13 W., W.M., at an elevation ranging from about 2000 to 2500 feet.

Ownership

Nine association placer claims of 160 acres each, known as the Red Gold Association nos. 1 to 9, are held by Carl Smedberg and associates of Gold Beach, Oregon.

Topography and climate

Red Flat is not a large nearly flat area as might be assumed from the name. However, as compared to most of the surrounding area, which is rugged and steep, it is relatively flat and undissected with a relief of two or three hundred feet. Scattered trees cover part of the area but large patches covered only by ground shrubs are common. The climate of the area is characterized by a rainy winter and a comparatively dry summer. Red Flat is well below the summit ridges which are remnants of the Klamath peneplain at an elevation of about 3500 feet.

Development

A few shallow trenches and a shaft reported by Morrison[3] to have been 32 feet deep, but now caved, comprise the bulk of the development work. A spring at the camp, when visited in June, had sufficient volume for all domestic needs and could probably supply a small mill also. Flycatcher Spring, about half a mile north of the camp junction, has a much smaller flow. The area is drained on the east by the North Fork of Pistol River, and on the west by the Big South Fork of Hunter Creek. Both stream channels lie several hundred feet below the level of the camp site.

Camp facilities include a cookhouse, bunkhouse, repair shed and other small buildings, some of which were under construction. A new sawmill and assay laboratory are located just below the camp buildings. The sawmill is used to provide lumber for construction of camp buildings. The timber is obtained from an adjacent stand of Port Orford cedar.

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[3] Morrison, J.E., Red Flat placers: Oregon Metal Mines Handbook, Coos, Curry, and Douglas Counties: State Dept. of Geology and Min. Industries Bull. 14-C, vol. I, p. 64, 1940.

Geology

As noted by Morrison, the deposit is of residual origin and was derived by the weathering in place of ultramafic rocks which underlie Red Flat and crop out on it in several places. These rocks, largely peridotite, intrude an older dark-colored greenstone (?) which occurs as isolated masses in some of the outcrops of peridotite. Greenstone (?) also crops out in a few places both just east and west of Red Flat proper. It carries quartz veinlets which are not present in the peridotite, suggesting that the greenstone (?) underwent one period of quartz mineralization that the peridotite did not.

The peridotite is one of the ultramafic intrusives which are common in southwestern Oregon and northern California. According to Wells[4] these rocks intrude all formations older than the Cretaceous including the Galice and Dothan formations of Jurassic age. A sample of peridotite from one of the outcrops on Red Flat was made up largely of olivine and darker green derived serpentine. The olivine was largely free of inclusions but magnetite grains were common in the serpentine. The greenstone (?) in the Red Flat area may be similar in age to or possibly older than the Galice and Dothan formations. The report by Butler and Mitchell[5] contains a geologic sketch map of Curry County which shows both the Dothan formation and the Colebrooke schist in contact with peridotite in the Red Flat area. According to Maxson[6], acid intrusives of late Jurassic or early Cretaceous age intrude the ultramafic rocks elsewhere in the Klamath Mountains.

The Red Flat surface was probably formed by an erosional cycle subsequent to that which formed the Klamath peneplain.[7] This is supported by Diller's statement that although residual deposits may have covered the Klamath peneplain, they have been largely removed. The Klamath peneplain, according to Diller, was developed while the Wimer beds were being laid down in the sea in the adjacent area southwest of the drainage of the Trinity River at the northern end of the Coast Range of California. As the Wimer beds, on the edge of the plateau at an elevation of about 2200 feet, 13 miles northeast of Crescent City, California, were then considered to be of Miocene age, Diller assigned the formation of the Klamath peneplain to that epoch. However, Diller pointed out that although the deposition of the Wimer beds occurred in late Tertiary time and probably in the late Miocene, further study might show that the Wimer beds and correlative formations are of Pliocene or even Pleistocene age and hence the age of the Klamath peneplain would be correspondingly reduced. Unfortunately no further attempt to date the Wimer fauna is known to have been made. Hence the development of the Red Flat surface, which apparently took place after the Klamath peneplanation, probably occurred during Pliocene time. Laterization of the Red Flat surface occurred during or subsequent to that epoch and prior to the elevation of the Red Flat surface to its present position. The uplift probably began late in the Pliocene or early Pleistocene and the ensuing erosion has removed part of the lateritic cover from Red Flat.

The laterite

The name Red Flat was undoubtedly suggested by the reddish color of the residual soil or laterite which covers much of the flat. Outcrops of country rock are fairly common. Color of the laterite ranges from yellow through yellowish brown and brown to deep reddish brown, and the texture is soft and earthy or mealy when dry. However, the moist laterite from the drill holes tended to be darker and mottled in places; some of the samples obtained

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4 Wells, F.G., Preliminary geologic map of the Grants Pass quadrangle, Oregon: State Dept. of Geology and Min. Industries, 1940.

5 Butler, B.S., and Mitchell, G.J., Preliminary survey of the geology and mineral resources of Curry County, Oregon: Oregon Bur. Mines and Geology, Min. Resources of Oregon, vol. 2, no. 2, October 1916.

6 Maxson, J.H., Economic geology of Del Norte and Siskiyou Counties, northwesternmost California: California Jour. Mines and Geology, vol. 29, nos. 1 and 2, January and April 1933.

7 Diller, J.S., Topographic development of the Klamath Mountains, U.S. Geol. Survey Bull. 196, 1902.

were quite plastic. As shown in roadcuts and by auger drilling, unaltered or only partially altered ultramafic rock ranging in size from small pieces to boulders occur scattered through the laterite. As a result some of the laterite in the drill holes was quite gritty. The surface of the laterite as well as that of outcrop areas is in places covered by numerous hard round "shots" or concretions commonly 1/8 to 1/4 inch in diameter. The areal extent of the lateritic soil is limited by that of the flat itself but the maximum thickness is not known. None of the four drill holes went deeper than 11 feet. However, the laterite may actually be thicker in places for it is believed that the peridotite rocks encountered at the bottom of three of the holes were loose. A caved shaft a short distance north of the mining camp near the south end of Red Flat is reported to have penetrated somewhat more than 32 feet of lateritic material. Thus the thickness of the laterite ranges from nil where the country rock crops out to at least 32 feet in places.

The chemical composition of the laterite is shown by the following analyses. Sample P-5452 is a composite of the samples from hole 1, and sample P-5455 is a composite of samples from hole 4. Sample P-5455 contained numerous rock fragments which account for a much higher silica and a lower iron content.

	P-5452 (composite of hole 1)	P-5455 (composite of hole 4)
Fe	42.51 %	22.36 %
Al_2O_3	10.76	16.80
SiO_2	7.58	24.49
Cr_2O_3	3.31	1.53
Ni	0.845	0.516
TiO_2	0.75	1.16
Au	0.015 oz/ton	0.004 oz/ton
Pt group metals (determined spectrographically)	nil	nil

Analyses of the samples from the 4 auger holes are shown graphically on following pages. Most of the samples contained only a trace of gold or silver but two samples did contain 0.02 ounce gold and a third contained 0.20 ounce silver. Although mercury has been reported to occur in the laterite, none was present in any of the samples collected by the Department.

As shown by these analyses the nickel content ranges from 0.27 to 1.46 percent. Partial chemical analyses of successive pan concentrates, tailings, and slimes of a composite sample of material from holes 1 and 4 are given on the following page. They indicate that the nickel content of the lighter fractions is much greater than that of the concentrate.

Petrographic examination of the laterite from hole no. 1 (sample P-4838) failed to reveal the nickel-bearing mineral or minerals. Most of the laterite is made up of limonite, largely goethite, and magnetite grains. The magnetite grains, which are residual from the weathering of the peridotite, contain a small amount of nickel. Chemical analysis shows that most of the nickel in sample P-4838 occurs in the slimes which are largely limonite. Other constituents of sample P-4838, which together contain only a small percentage of nickel, are serpentine (chrysotile), opal (including associated chalcedony), chromite, and a mica-like mineral. The last is fairly common and is similar in most properties to phlogopite but differs in that, unlike the micas, it has an optically positive interference figure.

Chemical analysis of a selected portion of the slimes (sample P-5594 of panning test) showed that it contained 0.74 percent nickel, 4.70 percent magnesia, 13.88 percent silica, and 65.88 percent R_2O_3 (mainly iron oxides). Petrographic examination of this selected slimes sample showed it to be predominantly limonite. No nickeliferous minerals were identified. A sample of soft limonite from Nickel Mountain near Riddle, Oregon, carefully collected by Pecora and Hobbs[8] in 1942, was reported to contain 1.3 percent nickel.

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[8] Op. cit.

Thus the analyses indicate that most of the nickel, as well as silica, is intimately associated with the limonite. Possibly the nickel is tied up molecularly with the limonite.

Results of Panning Tests, Red Flat Laterite
Composite of Drill Holes 1 and 4

Head sample
800 grams

↓

1st concentrate

← P-5589
Nonmagnetic fraction
(28 grams - 3.5%)
0.166% Ni
22.77 Cr_2O_3
21.20 Fe

→ P-5590
Magnetic fraction
(25.7 grams - 3.21%)
0.17% Ni
14.58 Cr_2O_3
43.91 Fe

↓

P-5591
2nd concentrate
(16.0 grams - 2.0%)
0.277% Ni
8.25 Cr_2O_3
27.45 Fe

P-5594
Slimes
(495.6 grams - 61.95%)
0.711% Ni
1.11 Cr_2O_3
35.90 Fe
14.42 SiO_2

↓

P-5592
3rd concentrate
(19.7 grams - 2.46%)
0.374 % Ni
5.21 Cr_2O_3
26.35 Fe

↓

P-5593
Sand tails
(215 grams - 26.87%)
0.556% Ni
1.35 Cr_2O_3
29.46 Fe
18.36 SiO_2

Loss of slimes and sand during panning amounted to 10.48 percent.

The lateritic origin of the deposit is indicated by the presence of residual magnetite, chromite, and serpentine as well as by pieces of peridotite.

Derivation of the laterite from the peridotite is also shown by the following spectrographic analyses. Sample P-4848 is a specimen of peridotite exposed near hole 2, and sample P-4827 is laterite from hole 1. Also listed in the table is the spectrographic analysis of the concretions or "shots", sample P-4849, which occur on the surface near hole 2.

Table 1

Percentage	P-4848 (peridotite exposed near hole 2)	P-4827 (laterite from hole 1. 0' to 1'2")	P-4849 (concretions from surface)
More than 10%	iron silicon magnesium	iron	iron aluminum (Chem., 19.92% Al_2O_3)
1 - 10%	----	silicon aluminum	silicon (Chem., 9.38% SiO_2)
0.1-1%	aluminum calcium	magnesium chromium cadmium nickel (Chem., 0.33%)	magnesium chromium cadmium sodium nickel (Chem., 0.178%)
0.01 - 0.1%	chromium nickel (Chem., trace) cadmium vanadium manganese sodium	calcium vanadium manganese sodium titanium boron tin	calcium vanadium manganese titanium
0.001 - 0.01%	titanium boron tin copper cobalt zirconium strontium potassium	copper cobalt zirconium strontium potassium	boron copper cobalt zirconium strontium potassium tin molybdenum
Less than 0.001%	barium silver molybdenum	barium silver molybdenum	barium

All of the elements shown to be present in the laterite occur in the peridotite. The spectrographic analyses show that of the major constituents of the peridotite the laterization removed much of the magnesium and part of the silicon, and concentrated the iron. The weathering process also concentrated the nickel and aluminum. Chemical analyses show that the peridotite near hole 2 contains a trace of nickel whereas a composite sample of the laterite from hole 2 contains 0.796 percent nickel. A sample of the greenstone, P-4860, from an outcrop northeast of hole 1 did not contain any nickel.

As previously noted, the weathering which produced the laterite probably occurred during the Pliocene before the area was elevated to its present position. Erosion has already removed part of the laterite as indicated by the presence on outcrop areas as well as on the surface of the laterite of hard round "shots" or concretions which were uncommon in the samples from the auger holes. The spectrographic analysis of a sample, P-4849, of the "shots" from the surface near hole 2, given in table 1, shows a further reduction in the magnesium content and a further increase in the aluminum content. Chemical analysis of this sample shows that the "shots" contain 19.92 percent Al_2O_3 (and 9.38 percent SiO_2) or nearly twice as much alumina as in the laterite from hole 1. The nickel content of this sample was 0.178 percent and that of the "shots" from 30 feet west of the road and 0.19 mile south of hole 1 was 0.341 percent.

From the amount of sampling done, there appears to be a tendency for the nickel content to increase with depth. If the concretions represent a former higher horizon, their lower nickel content follows the apparent trend. The apparently greater nickel content of the lower part of the lateritic section may be the result of greater leaching of the nickeliferous material in the upper part of the section or it may be due to an enrichment from above. Possibly both processes are responsible. However, if the nickeliferous minerals are formed near the bottom of the lateritic section from the olivine in the peridotite at the same time as the limonite, as seems reasonable, the lower nickel content of the upper part of the laterite may be largely the result of leaching. Possibly prospecting at depth will show appreciable enrichment.

The Red Flat deposit is similar in several respects to the brick-red laterite soil at Nickel Mountain[9] which ranges in thickness from a thin veneer to 9 feet. Three composite samples of the Nickel Mountain laterite were reported to contain 0.95, 1.02, and 1.10 percent nickel, respectively. Pecora and Hobbs stated that the mineral in the soil containing the nickel is not known.

The Department plans to make further studies of Oregon nickel-bearing laterite.

A graphic representation of drill-hole sampling results follows:

Hole No. 1

Sample number	Depth feet		Assay Au oz.	Ag oz.	Ni %	Hg %
	0					
P-4827	1		nil	tr	0.33	---
P-4828	2		0.02	tr	0.362	nil
P-4829	3		---	---	0.29	---
P-4830	4		nil	tr	0.59	nil
P-4831	5		tr	tr	0.695	nil
P-4833	6		tr	tr	0.959	nil
P-4834	7		nil	nil	1.09	nil
P-4835	8		0.02	tr	1.25	nil
P-4837	9		tr	nil	1.34	nil
P-4838	10		nil	nil	1.46	nil
P-4839	11	Bottom	---	---	1.18	---

9 Pecora, W. T., and Hobbs, S. W., op. cit.

Hole No. 2

Sample number	Depth feet		Assay Au oz.	Ag oz.	Ni %	Hg %
	0					
P-4841	1		nil	nil	0.46	---
P-4842	2		---	---	1.38	---
P-4843	3		nil	nil	0.65	nil
P-4844	4		tr	tr	0.62	---
P-4845	5		---	---	0.62	---
P-4846	6		nil	tr	1.007	nil
P-4847	7	Bottom	nil	tr	1.29	---

Hole No. 3

Sample number	Depth feet		Assay Au oz.	Ag oz.	Ni %	Hg %
	0					
P-4851	1		nil	0.20	1.17	---
P-4852	2		nil	tr	0.934	nil
P-4853	3		nil	tr	0.857	nil
P-4854	4		---	---	1.02	---
P-4855	5		nil	tr	1.14	---
P-4856	6		nil	tr	1.129	nil
P-4857	7	Bottom	nil	tr	1.25	---

Hole No. 4

Sample number	Depth feet		Assay Au oz.	Ag oz.	Ni %	Hg %
	0					
P-4861	1		nil	nil	0.357	nil
P-4862	2		nil	nil	0.585	nil
P-4863	3		---	---	0.406	---
P-4864	4		---	---	0.27	---
P-4865	5		nil	nil	0.516	nil
P-4866	6		---	---	0.605	---
P-4867	7		---	---	0.69	---
P-4868	8	Bottom	---	---	0.772	---

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OREGON CHROME REOPENED

Mr. W. S. Robertson has resumed work at the Oregon Chrome mine on the Illinois River, Josephine County, which had been closed down for nearly a year. Six men are employed - 3 shifts, 2 men to the shift-in driving a 500-foot drainage and access tunnel.

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TEXACO OIL TEST

On March 19 Texaco's test well, Clark and Wilson No. 6-1, was drilling ahead at 6500 feet.

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BILL SUSPENDS COPPER IMPORT TAX

HR 2404 which suspends the copper import tax until March 31, 1949, passed the House of Representatives March 12, 1947. The bill is now under consideration by the Senate Finance Committee.

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METAL MARKET PRICES

Copper has advanced to 21½¢ a pound, Connecticut Valley; lead to 15¢ a pound, New York. Zinc remains steady at 10½¢, East St. Louis. Foreign silver advanced early in the month from 75-3/4¢ to 86-1/4¢ on orders for export to India. However, the Reserve Bank of India issued an order prohibiting imports of silver, and the price receded to 77¢. Price of silver mined in the United States is fixed by law at 90½¢ an ounce. The quicksilver market has strengthened somewhat with quotations from $87 to $90 a flask, but trading has been moderate because of the usual uncertainties.

According to the Engineering and Mining Journal, New York, the advance in the price of copper and lead is caused by the heavy demands both here and abroad. Domestic prices were advanced to make them equivalent to foreign prices. The St. Joseph Lead Company issued a statement to the effect that the company raised its price reluctantly and felt that it is a mistake to make the price for the larger tonnages of lead consumed in this country the same as for the relatively small tonnages of foreign lead.

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CLEARING HOUSE

CH-93: WANTED - deposits of white potash feldspar. Philip S. Hoyt, 1002 Mills Bldg., El Paso, Texas.

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HENDRYX REAPPOINTED

Governor Earl Snell has reappointed Mr. H. E. Hendryx, Baker, as a member of the Governing Board of the State Department of Geology and Mineral Industries. Mr. Hendryx was named for a four-year term beginning March 19, 1947, and ending March 16, 1951. The appointment was confirmed by the State Senate March 18.

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MISCELLANEOUS PUBLICATIONS

	Price postpaid
THE ORE.-BIN: issued by the staff as medium for news about the Department, mines, and minerals. Subscription price per year	$ 0.25
Oregon mineral localities map (22 x 34 inches) 1946	0.10
Oregon quicksilver localities map (22 x 34 inches) 1946	0.25
Landforms of Oregon: a physiographic sketch, (17 x 22 inches) 1941	0.10
Index to topographic mapping in Oregon, 1946	Free
Index to geologic mapping in Oregon, 1946	Free

PUBLICATIONS

State Department of Geology and Mineral Industries, 702 Woodlark Building, Portland 5, Oregon

ETINS		Price postpaid
	Mining laws of Oregon, 1942, rev. ed., contains Federal placer mining regulations	$ 0.20
	Progress report on Coos Bay coal field, 1938: F.W.Libbey	0.10
	Geology of part of the Wallowa Mountains, 1938: C.P.Ross	0.50
	Quicksilver in Oregon, 1938: C.N.Schuette	0.50
	Geological report on part of the Clarno Basin, 1938: Donald K. MacKay	(out of print)
	Prelim. report on some of the refractory clays of western Oreg., 1938: Wilson & Treasher	(out of print)
	The gem minerals of Oregon, 1938: H.C.Dake	(out of print)
	Feasibility of steel plant in lower Columbia area, revised edition, 1940: R.M.Miller	0.40
	Chromite deposits in Oregon, 1938: John Eliot Allen	0.50
	Placer mining on Rogue River in relation to fish and fishing, 1938: H.B.Ward	(out of print)
	Geology and mineral resources of Lane County, Oregon, 1938: Warren D. Smith	0.50
	Geology and physiography of northern Wallowa Mts., 1941: W.D.Smith, J.E.Allen, et al	0.65
	First biennial report of the Department, 1937-38	(out of print)
	Oregon metal mines handbook: by the staff	
	A. Baker, Union, and Wallowa counties, 1939	(out of print)
	B. Grant, Morrow, and Umatilla counties, 1941	0.50
	C. Vol. I, Coos, Curry, and Douglas counties, 1941	(out of print)
	Vol.II, Section 1, Josephine County, 1942	(out of print)
	Section 2, Jackson County, 1943	0.75
5.	Geology of Salem Hills and North Santiam river basin, Oreg., 1939: T.P.Thayer (map only)	0.25
6.	Field identification of minerals for Oregon prospectors and collectors, second edition, 1941: compiled by Ray C. Treasher	(out of print)
7.	Manganese in Oregon, 1942: by the staff	0.45
8.	First aid to fossils, or what to do before the paleontologist comes, 1939: J.E.Allen	0.20
9.	Dredging of farmland in Oregon, 1939: F.W.Libbey	(out of print)
0.	Analyses and other properties of Oregon coals, 1940: H.F.Yancey & M.R.Geer	(out of print)
1.	Second biennial report of the Department, 1939-40	Free
3.	Investigation of reported occurrence of tin at Juniper Ridge, Oreg., 1942: Harrison & Allen	0.40
24.	Origin of the black sands of the coast of southwestern Oregon, 1943: W.H.Twenhofel	0.30
25.	Third biennial report of the Department, 1941-42	Free
26.	Soil: Its origin, destruction, and preservation, 1944: W.H.Twenhofel	0.45
27.	Geology & coal resources of Coos Bay quad., 1944: John Eliot Allen & Ewart M. Baldwin	1.00
28.	Fourth biennial report of the Department, 1943-44	Free
29.	Ferruginous bauxite deposits in N.W.Oregon, 1945: F.W.Libbey, W.D.Lowry, & R.S.Mason	1.00
30.	Mineralogical and physical composition of the sands of the Oregon coast from Coos Bay to the mouth of the Columbia River, 1945: W.H.Twenhofel	0.35
31.	Geology of the St. Helens quadrangle, 1946: W.D.Wilkinson, W.D.Lowry, & E.M.Baldwin	0.45
32.	Fifth biennial report of the Department, 1945-46	Free

G.M.I. SHORT PAPERS

1.	Preliminary report upon Oregon saline lakes, 1939: O.F.Stafford	0.10
2.	Industrial aluminum - a brief survey, 1940: Leslie L. Motz	0.10
3.	Adv. report on some quicksilver prospects in Butte Falls quad., Oreg., 1940: W.D.Wilkinson	(out of print)
4.	Flotation of Oregon limestone, 1940: J.B.Clemmer & B.H.Clemmons	0.10
5.	Survey of nonmetallic mineral production of Oregon for 1940-41: C.P.Holdredge	0.10
6.	Pumice and pumicite, 1941: James A. Adams	(out of print)
7.	Geologic history of the Portland area, 1942: Ray C. Treasher	0.15
8.	Strategic & critical minerals, a guide for Oregon prospectors, 1942: Lloyd W. Staples	(out of print)
9.	Some manganese deposits in the southern Oregon coastal region, 1942: Randall E. Brown	0.10
10.	Investigation of Tyrrell manganese and other nearby deposits, 1943: W.D.Lowry	0.15
11.	Mineral deposits in region of Imnaha and Snake rivers, Oregon, 1943: F.W.Libbey	0.15
12.	Preliminary report on high-alumina iron ores in Washington County, Oregon, 1944: F.W.Libbey, W.D.Lowry, & R.S.Mason	0.15
13.	Antimony in Oregon, 1944: Norman S. Wagner	0.15
14.	Notes on building-block materials of eastern Oregon, 1946: Norman S. Wagner	0.10
15.	Reconnaissance geology of limestone deposits in the Willamette Valley, Oreg., 1946: J.E.Allen	0.15
16.	Perlite deposits near the Deschutes River, southern Wasco County, Oreg., 1946: J.E.Allen	0.15
17.	Sodium salts of Lake County, Oregon, 1947: Ira S. Allison & Ralph S. Mason	0.15

Hole No.	O.B. Thickness	Laterite Thickness	Laterite Grade	Depth to Sapprolite	Sapprolite Thickness	Sapprolite Grade
1	—	—	—	15.0	10.0	1.04
2	—	—	—	—	—	—
3	—	—	—	30.0	15.0	1.18
4	—	10.0	1.24	—	—	—
5	—	—	—	—	—	—
6	—	—	—	—	—	—
7	—	—	—	5.0	20.0	1.21
8	—	—	—	—	—	—
9	—	10.0	1.16	10.0	5.0	1.20
10	—	10.0	1.08	10.0	15.0	1.15
11	—	—	—	15.0	25.0	1.00
12	—	10.0	0.90	10.0	5.0	1.00
13	—	—	—	25.0	5.0	1.10
15	—	5.0	1.67	5.0	5.0	1.32
16	—	5.0	0.92	—	—	—
17	5.0	10.0	1.25	—	—	—
18	—	—	—	—	—	—
19	—	—	—	—	—	—
20	—	—	—	—	—	—
21	—	—	—	—	—	—
22	—	—	—	—	—	—
23	—	5.0	1.16	5.0	20.0	1.63

Sample No.	Interval	Ni	Ni weighted	Co	Co weighted
1	7	0.69	4.83	0.10	.7
2	6	0.50	3.00		
3	6	0.42	2.52		
4	1	1.12	1.12		
5	7	0.74	5.18		
6	4	0.70	2.8		
7	6	0.58	3.48		
8	7	0.94	6.58		
9	2	0.45	.9		
10	12	0.83	9.96		
11	4	0.96	3.84		
12	3.5	0.90	3.15		
13	2	0.41	.82		
14	13.5	0.86	11.61		
15	2	0.55	1.1		
16	6.6	0.34	2.24		
17	9	0.69	6.21		
18	5	0.90	4.5		
19	5.5	0.78	4.29		
20	8	1.11	8.88		
21	4.5	1.~~55~~ 54	6.93		
22	2	0.13	.26		
23	2.5	0.51	1.28		
23 ~~24~~	4.2	0.18	.76		
24 ~~25~~	5	0.49	2.45	0.08	.4
25 ~~26~~	6.5	0.72	4.68	0.06	.39
26 ~~27~~	5.6	1.06	5.94		
28	9.5	0.63	5.99	0.11	1.045

Sample No.	Interval	Ni	Ni Weighted	Co	Co Weighted
29	9.5	0.61	5.80		
30	10.0	.65	6.50		
31	25.0	.78	19.5		
32	15.0	.57	8.55		
33	25.0	.83	20.75	Co total	
34	15	.66	9.9	interval	
35	20	.55	11.0	↓	
	276.4		197.3	28	2.54

Ni = 0.71 %

Co = 0.09 %

1

~~Feet of thickness~~ *
~~lbs / Ton~~ (10)
~~weighted~~

Sec 30 (Ore Block) ~~[illegible]~~

Hole No.	Interval	% Ni	% Co	% Combination	Ni lbs/ton	Co
1	15'-25'	0.92	0.12	1.04	18.4	2.4
* 2	15-25'	0.61	0.10 *	0.71	12.2	2.0
3	30'-50'	1.09	0.04	1.13	21.8	0.8
4	0-10'	1.24	0.10	1.34	24.8	2.0
* 5	15-25	0.65	0.10 *	0.75	13.0	2.0
* 6	5-20	0.64	0.10 *	0.74	12.8	2.0
7	5-25'	1.06	0.15	1.21	21.2	3.0
* 8	0-10'	0.48	0.10 *	0.58	9.6	2.0
9	0-20'	1.04	0.07	1.11	20.8	1.4
10	0-25'	1.04	0.08	1.12	20.8	1.6
11	15-40'	1.00	0.13	1.13	20.0	2.6
12	0-15'	0.81	0.13	0.94	16.2	2.6
13	10-30'	0.89	0.02	0.91	17.8	0.4
15	0-10'	1.50	—	1.50	30.0	—
16	0-10'	0.79	0.27	1.06	16.8	5.4
17	5-20'	1.11	—	1.11	22.2	—
* 18	0-10'	0.50	0.10 *	0.60	10.0	2.0
* 19	5-15'	0.49	0.10 *	0.59	9.8	2.0
20	0-25'	0.95	0.10	1.05	19.0	2.0
* 21	0-10'	0.50	0.10 *	0.60	10.0	2.0
22	—	—	—	—	—	—
23	0-25'	1.36	0.20	1.56	27.2	4.0
	325'					

Average Grade unweighted 0.99% Ni + Co, weighted 1.04% Ni + Co

Average Depth or width or ore 15.5' (nothing less than 10')

* Low Grade

* Co not sampled, used 0.10% Co for ~~combination~~ ~~Ni + Co~~ calculations

Hole No	Weighted ft. lbs./Ton Ni	Co
1	184	24
2	122	20
3	436	16
4	248	20
5	130	20
6	192	30
7	424	60
8	96	20
9	416	14
10	520	40
11	500	65
12	243	39
13	356	8
15	300	—
16	158	54
17	333	—
18	100	20
19	98	20
20	475	50
21	100	20
22	—	—
23	680	100
	6111	640

Average Ni 6111/325 = 18.80 lbs ton x $3.00 = $56.40

Average Co 640/325 = 1.97 lbs ton x $10.00 = $19.70

$76.10

Last assessment work

Owner, Claim, Company

			Lead File
Sec 30 1982	PL	Red Flats Nickel, Curran, James M	17034
1980	PL	" " " , Spicer, M. John	19424
Sec 31 1982	PL	Big Basin Nickel Co, " " "	19418
1982	PL	Red Rock claim Spicer, M. John (co-ofmany	20954
1982	LD	Ocean View I, Peg Leg II, Gerard, Gerard the Drakatos, Joseph W; Drakatos, Frosinia; Drakatos Anna; Sherwood, Edward R.	44905

Big Basin Nickel Corp
P.O. Box T
17860 Gardner Ridge Rd
Brookings, Oregon 97415

Vera F Leeds, sec treas
James M. Curran Pres
Thorton C. Leeds +
Douglas T. Leeds did assessment work 82

M. John Spicer
Attorney at Law
140 Gauntlett
Gold Beach, Or 97444

(8:30 at office)
(503) 247-7003 247-7553
P.O. Box 645

Lawyer for Big Basin + Red Flats

Dennis
Mr. Winn
560 Shore Pines Ave
Coos Bay, Or 97420

President ?
Red Flats Nickel Corp
(might be good info source)

Lode
Frosinia Drakatos
100 Del Mar Rd
Coos Bay, Or 97420

Gold Dome Natural Resources, Inc.			Book	page
Lode — Ocean View	CRMC	44905	96	506
Peg Leg	"	6	"	7
Gerard	"	7	"	8
Gerard II	"	8	"	9

(756-0521) 8:30 call in morning

Jon D. Dowers
3720 Spruce
North Bend, Or 97459

756-6621
Call for info.

$N\frac{1}{2}$ $NW\frac{1}{4}$ $NW\frac{1}{4}$ sec 31, T 37 S, R 13 W
Ocean View - I, Lode claim

Red Flats Nickel Corp
1050 Stock Slough
Coos Bay Or 97420

Coastal Mining Co, Lessee of Big Basin Nickel Corp claims, is a subsidiary of Hanna Nickel

Claims cover area of large extent T 37 S R 13 W -
SW $\frac{1}{4}$ sec 7, W $\frac{1}{2}$ sec 18, sec 19, W $\frac{1}{2}$ sec 20,
sec 30, E $\frac{1}{2}$ sec 31, NW $\frac{1}{4}$ sec 32 - T 37 S R 14 W -
SE $\frac{1}{4}$ sec 12, N $\frac{1}{4}$ + SE $\frac{1}{4}$ sec 13, sec 24, sec 25

The above are all not in the lease, but most probably are?

Coastal Mining Company
333 South Carson Meadows Drive #44
Carson City, Nevada 89701
(702)-883-9651
Roger Gosh, Landman

Above takin from letter of Sept 17, 1981, they may not hold lease any more

Geology: The country rocks consist of a greenish and partially serpentinized sandstone conglomerate in contact with a steeply dipping layer of shale. Both are probably of the "Myrtle formation" of Cretaceous age. The strike of the contact is approximately N. 35° E., and its dip is steeply northwest. Cinnabar occurs in the conglomerate immediately adjacent to the steeply dipping shale hanging wall. The cinnabar occurs both as disseminations and as thin coatings surrounding the pebbles. Traces of cinnabar can be found along the contact for more than 1,000 feet. An 8-foot channel sample from one of the cuts assayed 0.2 pounds of quicksilver to the ton. Another channel sample 5 feet in length assayed 3.05 pounds per ton. Another assayed 0.15 pounds per ton.

RED FLAT PLACERS AREA

Red Flat is a semi-level area adjacent to the North Fork of Pistol River in T. 37 S., R. 13 W., Curry County, about 17 miles by road east of Gold Beach. The map of the Gold Beach quadrangle gives the topography of the area.

Placer operations to recover both gold and quicksilver from the residual soils of the flat were begun in the early 1930's. Since that time, several large groups of placer claims have been located and there has been considerable prospecting activity. Concentrating equipment and small retorts have been erected, but there is no record of production. Present ownership and property boundaries are not accurately known. Organizations and individuals involved over the years have been The Red Flats Association or Red Gold Mining Co. headed by Mary Smedberg and J. A. Walsh, both of Gold Beach; Red Ridge Mining Co., represented by Harry Hedderley of Gold Beach; The Glade Creek Placer Association; and numerous others.

Most of Red Flat appears to be underlain by various types of peridotite which has been serpentinized to varying degrees. Red clayey soil, a product of weathering, interspersed with loose boulders of serpentine and peridotite, form the surface mantle. Since the terrain is fairly level, the products of weathering are only slowly removed. At least one prospect hole sunk to a depth of 30 feet failed to penetrate bedrock. Both cinnabar and native quicksilver occur locally in the soils of the flat but nowhere are representative samples known to have been taken that assayed more than a small fraction of a pound of quicksilver per ton. Parts of the deposit have been sampled for nickel by the department (Libbey and others, 1947) and by the U.S. Bureau of Mines (Hundhausen and others, 1954). Drill-hole samples tested for nickel were also assayed for quicksilver, but results were not favorable.

DIAMOND CREEK PLACERS AREA

The Diamond Creek placers are on the North Fork of Diamond Creek in the extreme southeastern corner of Curry County. The deposits, at an elevation of about 2,150 feet, extend southward along the creek into California. Those in Oregon lie in sec. 16, T. 41 S., R. 10 W., in the Chetco Peak quadrangle. Present ownership is unknown. The property was equipped with hydraulic equipment in 1929 and a small amount of ground has since been sluiced by various owners and lessees. Little work has been done for many years. The geology and development are described by Cater and Wells (1953, p. 126-7) as follows:

> "The cinnabar is scattered along fine joint fissures in a mass of propylitized diorite. The feldspar in this rock has been completely altered to kaolinite and sericite, and the amphibole has been altered to limonite. The altered diorite is exposed over the top of the ridge west of the camp and farther west is in contact with, or continuous with, the dikes passing near the Sunny Brook prospect. A tongue of serpentine crops out to the north, and to the east less altered rocks occur, including either a fresh hornblendite or gabbro containing inclusions of serpentine.
>
> "The original locator, John Griffin, dug a ditch along the top of the ridge and ground-sluiced what was originally a small slide. The water was run through a 10-inch sluice box equipped with Hungarian block riffles. The concentrates were retorted in two 4-inch pipes. Later equipment, installed by the J. I. L. Dredging Co. of Spokane, which leased the property was essentially a refinement of the above. A 3-inch giant was operated in the slide and the material run through a series of sluices in an attempt to concentrate the heavier cinnabar crystals by gravity separation. The process was extremely inefficient, operations were abandoned, and the property has been idle a number of years."

Coos, Curry, and Douglas Co.

RED FLAT PLACERS Gold Beach Area

Owners: Carl Smedberg and associates, Gold Beach, Oregon.

Location: The Red Flat Placers are located in secs.18,19, and 30, T.37 S., R.13 W., and secs.13 and 21, T.37 S., R.14 W., about 8 airline miles southeast of Gold Beach. The property can be reached by going south from Gold Beach to Pistol River, then taking Pyramid Rock Lookout road for sixteen miles, with increase of 2000 feet in elevation. Barometer elevation at property is 3500 feet.

Property: There are nine association placer claims of one hundred sixty acres each and known as the Red Gold Assocation Nos.1 to 9 inclusive.

Facilities: There is no timber on the claims, but plenty of timber is available a few miles to the west. There are a number of small springs on the Flat for domestic water, and plenty of water available in Hunter Creek and Pistol River for commercial use.

Geology: The Red Flat Placers are so named because the residual surface material is a bright red color, due to the large iron oxide content. The product of erosion on either side of the deposit is not red, so a fairly distinct boundary line is seen. This adjoining country rock is serpentine and there appears to be no distinctive difference in the rocks to the west or east of the deposit. The Red Flat material is a residual deposit, apparently with no outcrops of material in place. The terrain being fairly level, the products of weathering are not carried away. The deepest prospect hole, now caved, is said to be thirty-two feet deep and all in loose material.

Sampling and Assaying: Carl Smedberg, one of the owners, stated that "fire assays would not recover the values; that a wet assay is required." Smedberg pointed out a good place to take samples, and one was taken after digging a hole three feet deep into the loose material. The few loose rocks were saved and assayed separately with values of 70¢ gold per ton. The red material taken from top to bottom of the hole assayed no values in gold or silver. Several hand specimens were taken of the serpentine and assayed, but no values in gold or silver were found. It is reported that the operators of Red Flat Placers have their samples assayed by a wet method by Paul Smith Laboratories of 9315 Pico Blvd., Los Angeles, Cal. These assays, dated July 18 and Sept. 9, 1933, claim $20.00 in gold, $6.00 silver, $20.00 nickel, and $20.00 in platinum and allied metals A sample of black sand from this locality submitted by Mr. T. Edwards assayed 40.9 percent chromic oxide and 0.055 percent mercury.

Informant: J. E. Morrison, 37.

SIGNAL BUTTE GROUP (Chromite) Gold Beach Area

Located in secs.25, 30, T.36 S., R.14 W., and secs.28, 29, 30, 31, 32, T.36 S.,R.13 W., and sec.1, T.37 S.,R.14 W., and secs. 5,6, T.37 S., R.13 W.

"Thirty-five claims, scattered over about 5 square miles around Signal Buttes, were visited and several of them studied in detail.

"The Signal Buttes district is one of considerable geologic complexity. The serpentines were intruded into a series of shales, sandstones, banded cherts and quartzites. The dense highly siliceous members such as cherts and quartzites seem to have been left relatively unaltered; sandstones were replaced in all degrees, and shaly sediments seem to have been more or less completely replaced over large areas with only occasional remaining patches, now altered to schist. The "buttes" themselves are later volcanic plugs and

Oct 19, 1983 Red Flat

Sample # 11979, Qtz float along side of logging road in sec 31? east side S/red flat timber sale.

Oct 20, 1983 Red Flat

Sample # 11980, starting at main intersection of roads in sec 30, 1/10 mile north in trench below road, 8' cut in middle of trench, red latterite soil

Sample # 11981, intersection + 3/10 mile N in trench on lower side of road, 7' cut red latterite soil

Sample # 11982, intersection + 5/10 mile N in trench on lower side of road 7' cut red latterite soil.

serpentine, near nine mile marker, two dozer trenches above road

sample # 11988, intersection + 1 7/10 miles N 6' cut in west bank blue grn serpentine above culvert

sample # 11989, directly above # 11988 7' cut in mostly transported latterite material, steep slope latterite cover thin except in draws.

sample # 11990, intersection + 1 9/10 miles N along road north bank two ft. cut in mostly residual latterite soil

sample # 11991, intersection + 1/10 mile S 3' cut on east bank of road, qtz float in soil md orange rd soil, qtz str in foliation planes of underlying rk, up to 2 cm thick, rk is gy schist.

Sample # 11983, intersection + 7/10 mile — N near ~~sec line 30 + 19~~, 8' road cut sample on upper side of road in latterite soil just above culvert —

sample # 11984, intersection + 9/10 mile N on upper west bank of road by culvert 5' cut, latterite soil

sample # 11985, intersection + 1 1/10 mile N on upper west bank of road just N of culvert that Hg was found. Top 1.5' red latterite soil, bottom 5.5' lt blu gy serpentine mixed w/ latterite, becomming more serpentine towards bottom.

sample # 11986, intersection + 1 3/10 mile N, 7' cut upper west side of road 50% latterite soil, 50% altered blu gy serpentine

Sample # 11987, intersection + 1 5/10 mile N, 10' cut on upper west bank of road, latterite + altered blu gy

Sample # 11992, intersection + 3/10 mile — S, 1.5' cut on east bank in soil w/ qtz float, nearby outcrop lt blu gy schist.

sample # 11993 intersection + 5/10 mile S 7' cut on east bank in mostly soil — w/ qtz float, lt blu gy schist rks nearby have qtz str + blebs. Between samples 11992 + 11993 latterite soils cover ground.

sample # 11994, intersection + 7/10 mile S near Gold Dome road to shaft, 6' cut in overburden on west bank rk adjacent qtz diorite w chlorite. no visible qtz float.

sample # 11995, intersection + 9/10 mile S, cut on east bank 3' in overburden, minor qtz float, latteritic soil, sample 100' N of road spur that sample # 11979 was taken on

- sample #11996, intersection + 1 $\frac{1}{10}$ miles S, on West bank, 5' cut in latite soil, buff gy serpentine adjacent

- sample # 11997, intersection + 1 $\frac{3}{10}$ miles S on east bank 5' cut in latite soil,

- timber larger

r Valley Laboratories
Box 926 -- 308 N. TAYLOR
RN, IDAHO 83849
(208) 556-1593

SUNSHINE MINING COMPANY
c/o EXPLORATION DEPARTMENT
BOX 1080
KELLOGG, IDAHO 83837

EMBER 5,1983 SUSH0701.297 ATTN: GEORGE SINTAY

EST FOR:	Au	Ag	Au	Ag	Au	Ag	Hg
METHOD USED:	Geo Chem	Geo Chem	Geo Chem	Geo Chem	Amalgamation	"	Assay
RESULTS IN:	ppb	ppm	ppb	ppm	oz/ton	oz/ton	ppm
			Aqua	Regia			
#11971	<10	.1	<100	<.1	.002	.04	N/R
#11972	10	.1	<100	.1	.001	.09	N/R
#11973	13	.1	<100	<.1	.003	.05	N/R
#11974	17	.1	<100	.1	<.001	.11	N/R
#11975	20	2.0	480	.1	NIL	.10	N/R
#11976	<10	<.1	<100	<.1	.001	.14	N/R
#11977	28	1.4	<100	.1	.001	.09	N/R
#11978	<10	.8	<100	<.1	NIL	.06	N/R
#11979	<10	<.1	106	<.1	N/R	N/R	.059
#11980	16	.4	<100	.1	N/R	N/R	.154
#11981	<10	.6	<100	.1	N/R	N/R	.204
#11982	<10	.2	<100	<.1	N/R	N/R	.127
#11983	<10	.2	<100	<.1	N/R	N/R	.109
#11984	<10	1.0	<100	.1	N/R	N/R	.134
#11985	<10	1.2	<100	<.1	N/R	N/R	.050
#11986	<10	.6	<100	<.1	N/R	N/R	.044
#11987	16	.2	<100	<.1	N/R	N/R	.043
#11988	<10	.4	<100	<.1	N/R	N/R	.024
#11989	22	<.1	<100	.1	N/R	N/R	.070
#11990	<10	.2	<100	<.1	N/R	N/R	.085
#11991	36	<.1	<100	<.1	N/R	N/R	.168
#11992	<10	<.1	<100	<.1	N/R	N/R	.106
#11993	<10	<.1	<100	<.1	N/R	N/R	.194
#11994	24	<.1	<100	<.1	N/R	N/R	.029
#11995	<10	1.0	266	.1	N/R	N/R	.102
#11996	<10	<.1	<100	.2	N/R	N/R	.166
#11997	<10	.4	<100	<.1	N/R	N/R	.058

PAGE 1 OF 1 CHARGES: $708.00

Wayne Sorensen
WAYNE SORENSEN, Chief Chemist

Table 5. Sample assay results, Red Flat area

Sample number ****	Assay number	Type	Depth interval (ft)	Location ¼	Location Sec.	Ni	Percent Co	Percent Cr	Percent Fe
1	AJG-69	Auger	0- 7	S/NE	13	0.69	0.10	----	----
2	AJG-68	Auger	0- 6	S/NE	13	0.50	----	----	----
3	5-AGG-80-84	Auger	0- 6	NW/SE	13	0.42	----	----	----
4	6-AAG-85	Auger	2- 3	N/SE	13	1.12	----	----	----
5	1-AGG-56-62	Auger	0- 7	NE/SE	13	0.74	----	----	----
6	2-AGG-63-66	Auger	0- 4	NE/SE	13	0.70	----	----	----
7	3-AGG-67-72	Auger	0- 6	SE	13	0.58	----	----	----
8	4-AGG-73-79	Auger	0- 7	SE	13	0.94	----	----	----
9	9-AAG-88	Auger	2- 4	E/SE	13	0.45	----	----	----
10	4-NR-2-15*	Channel	4-18	SE	13	0.83	----	----	----
11	7-AAG-86	Auger	2- 6	SE	13	0.96	----	----	----
12	8-AAG-87	Auger	2- 5.5	E/SE	13	0.90	----	----	----
13	3-NR-2-14*	Chip	0- 2	NW/SW	18	0.41	----	----	----
14	5-NR-2-16-20*	Auger	0-13.5	NW/SW	18	0.86	----	----	----
15	2-NR-2-13*	Channel	0- 2	SW	18	0.55	----	----	----
16	RF-4(5-15-75)***	Auger	0- 6.6	SE/SE	13	0.34	----	----	14.0
17	RF-7(5-76)***	Auger	0- 9	S/SE/SE	13	0.69	----	0.70	21.0
18	8-NR-3-1-2*	Auger	0- 5	SW	18	0.90	----	----	----
19	6-NR-2-21-22*	Auger	0- 5.5	NW/NW	19	0.78	----	----	----
20	7-NR-2-23-25*	Auger	0- 8.0	NW/NW	19	1.11	----	----	----
21	AJG-57	Auger	5- 9.5	NW	19	1.54	----	----	----
22	AKG-19	Channel	0- 2	W/NW/NW	25	0.13	----	1.00	----
23	AJG-11	Auger	0- 2.5	W/NW	25	0.51	----	----	----
23	RF-1(5-14-75)***	Auger	0- 4.2	NW/NW	25	0.18	----	----	----
24	AJG-12	Auger	0- 5	NW	25	0.49	0.08	0.39	12.5
24	RF-3(5-14-75)***	Grab	Creek cut	NE/NW	25	0.41	----	----	----
25	AJG-62	Auger	0- 6.5	NW/NE	25	0.72	0.06	----	----
26	AJG-63	Auger	0- 5.6	N/NE	25	1.06	----	----	----
27	AJG-14	Grab	Surface	W/SW/NW	25	0.75	----	----	----
27	RF-2(5-14-75)***	Grab	Surface	SW/NW	25	0.21	----	----	----
28	AJG-60-61	Auger	0- 9.5	E/NW	25	0.63	0.11	1.72	41.2
29	AJG-58-59	Auger	0- 9.5	E/NW	25	0.61	----	----	----
30	DH 6704**	Drill	0-10	NW	30	0.65	----	----	----
31	DH 6701**	Drill	0-25	SW/NW	30	0.78	----	----	----
32	DH 6705**	Drill	0-15	NW	30	0.57	----	----	----
33	DH 6702**	Drill	0-25	SE/NW	30	0.83	----	----	----
34	DH 6703**	Drill	0-15	NE/SW	30	0.66	----	----	----
35	DH 6706**	Drill	0-20	SE	30	0.55	----	----	----

* Appling (1955)
** Drill hole data furnished by Red Flats Nickel Corp; results averaged from 5-ft interval assays
*** Assayed by Hanna Mining Company
**** These numbers are found in Figure 5 and indicate locations from which samples were taken

8-22-83

Dennis Winn
Ned Pilato

>5200 Ac
R I - 5072

Unpat. Claims

0.8 Ni
0.14 Co
avg. - ore bed

Avg 12-ft depth -
from surface down...

Prim. Ni in rock

Dogami - drilled...
24 mit

Jan 1982
Contr. J0285021
Min. Sci. Div.
New Proced for Recover. Ni & Co
from Western ~~Laterite~~
A minerals research Contract rpt

Power - nearby
Power transformer - sitting @ Gold Beach

Robt Weldon - (poss. resigned -
move back to Spokane).

RED FLAT 1946

Sample number	Description	Au (oz)	Ag (oz)	Al_2O_3 (%)	SiO_2 (%)	Cr_2O_3 (%)	Ni (%)	Pt (oz)	Fe (%)	TiO_2 (%)	Mg (%)
P-4827	Hole no. 1 0'-1'	nil	Tr				.33				-
P-4828	Hole no. 1 1'-2'	.02	Tr				.362				Nil
P-4829	Hole no. 1 2'-3'						.29				
P-4830	Hole no. 1 3'-4'	Nil	Tr				.59				Nil
P-4831	Hole no. 1 4'-5'	Tr	Tr				.695				Nil
P-4832	Hole no. 1 5'-6'	Tr	Tr				.959				Nil
P-4834	Hole no. 1 6'-7'	Nil	Nil				1.09				Nil
P-4835	Hole no. 1 7'-8'	.02	Tr				1.25				Nil
P-4837	Hole no. 1 8'-9'	Tr	Nil				1.34				Nil
P-4838	Hole no. 1 9'-10'	Nil	Nil				1.46				Nil
P-4839	Hole no. 1 10'-11'	-	-				1.18				-
P-4841	Hole no. 2 0'-1'	Nil	Nil				.46				-
P-4842	Hole no. 2 1'-2'						1.38				-
P-4843	Hole no. 2 2'-3'	Nil	Nil				.65				Nil
P-4844	Hole no. 2 3'-4'	Tr	Tr				.62				-
P-4845	Hole no. 2 4'-5'						.62				
P-4846	Hole no. 2 5'-6'	Nil	Tr				1.007				Nil
P-4847	Hole no. 2 6'-7'	Nil	Tr				.129				-
P-4848	Rock exposed at Hole no. 2	Nil	Nil				Trace				Nil

RED FLATS

Sample number	Description	Au (oz)	Ag (oz)	Al_2O_3 (%)	SiO_2 (%)	Cr_2O_3 (%)	Ni (%)	Pt (oz)	Fe (%)	TiO_2 (%)	Hg (%)
				Assays							
P4849	Shots from Surface at Hole 2	Nil	Nil	19.92	9.38		.178				
P-4851	Hole no. 3 0'-1'	Nil	.20				1.17				-
P-4852	Hole no. 3 1'-2'	Nil	Tr				.934				Nil
P-4853	Hole no. 3 2'-3'	Nil	Tr				.857				Nil
P-4854	Hole no. 3 3'-4'						1.02				
P-4855	Hole no. 3 4'-5'	Nil	Tr				1.14				-
P-4856	Hole no. 3 5'-6'	Nil	Tr				1.129				Nil
P-4857	Hole no. 3 6'-7'	Nil	Tr				1.25				-
P-4860	Greenstone N.E. of Hole No.1	Nil	Nil				Nil				
P-4861	Hole no. 4 0'-1'	Nil	Nil				.357				Nil
P-4862	Hole no. 4 1'-2'	Nil	Nil				.585				Nil
P-4863	Hole no. 4 2'-3'	-	-				.406				-
P-4864	Hole no. 4 3'-4'	-	-				.27				-
P-4865	Hole no. 4 4'-5'	Nil	Nil				.516				Nil
P-4866	Hole no. 4 5'-6'	-	-				.605				-
P-4867	Hole no. 4 6'-7'	-	-				.69				-
P-4868	Hole no. 4 7'-8'	-	-				.772				-

Sample number	Description	Au (oz)	Ag (oz)	Al_2O_3 (%)	SiO_2 (%)	Cr_2O_3 (%)	Ni (%)	Pt (oz)	Fe (%)	TiO_2 (%)	Hg (%)
		Assays									
P-5452	Composite of drill hole 1	.015		10.76	7.58	3.53	.845	Nil	42.51	.75	Nil
P-5453	Composite of drill hole 2					2.01	.796	Nil			
P-5454	Composite of drill hole 3	.008				3.79	1.04	Nil			
P-5455	Composite of drill hole 4	.004	Tr	16.80	24.49	1.53	.516	Nil	22.36	1.16	
P-5589	Pan Concentrate of drill holes (composite of) 1 & 4 (non-magnetic fraction)					22.77	.166		21.20		
P-5590	Pan Concentrate of composite of drill holes 1 & 4 (magnetic fraction)					14.58	.170		43.91		
P-5591	Pan Concentrate of composite of drill holes 1 & 4 2nd concentrate.					8.25	.277		27.45		
P-5592	Pan Concentrate of composite of drill holes 1 & 4 3rd concentrate.					5.21	.374		26.35		
B-5593	Sand Tails composite of drill holes 1 & 4				18.36	1.35	.556		29.46		
P-5594	Slimes Composite of drill holes 1 & 4				14.42	1.11	.711		35.90		

RED FLAT LATERITE

Sample number		Sample width		Assay Au (ounces)	Assay Ag (ounces)	Assay Ni (percent)	Assay Hg (percent)
P-4827	0	1'-2"	Hole no. 1	Nil	Tr	.33	-
P-4828	1	1'-0"		.02	"	.362	Nil
P-4829	2	1'-0"				.29	
P-4830	3	1'-0"		Nil	Tr	.59	Nil
P-4831	4	1'-0"		Tr	Tr	.695	Nil
P-4833	5	1'-0"		Tr	Tr	.959	"
P-4834	6	1'-0"		Nil	Nil	1.09	"
P-4835	7	1'-0"		.02	Tr	1.25	"
P-4837	8	1'-0"		Tr	Nil	1.34	"
P-4838	9	1'-0"		Nil	"	1.46	"
P-4839	10	1'-0"		-	-	1.18	-
	11						
P-4841	0	1'-0"	Hole no. 2	Nil	Nil	.46	-
P-4842	1	1'-0"				1.38	-
P-4843	2	1'-0"		Nil	Nil	.65	Nil
P-4844	3	1'-0"		Tr	Tr	.62	-
P-4845	4	1'-0"				.62	
P-4846	5	1'-0"		Nil	Tr	1.007	Nil

				-	-	.69	-
P-4868	7	0'-10"		-	-	.772	-
	8						

Samphe number		Sample width		Assay Au (ounces)	Assay Ag (ounces)	Assay Ni (percent)	Assay Hg (percent)
	6						
P-4847		1'-0"	Hole no. 2	Nil	Tr	1.29	-
	7						
	0		Hole no. 3				
P-4851	0	1'-0"		Nil	.20	1.17	-
	1						
P-4852		1-0"		"	Tr	.934	Nil
	2						
P-4853		1'-0"		"	"	.857	"
	3						
P-4854		1'-0"				1.02	
	4						
P-4855		1'-0"		Nil	Tr	1.14	-
	5						
P-4856		1'-0"		"	"	1.129	Nil
	6						
P-4858		1'-0"		"	"	1.25	-
	7						
P-4861	0	1-0"	Hole no. #	Nil	Nil	.357	Nil
	1						
P-4862		1'-0"		"	"	.585	"
	2						
P-4863		1'-0"		-	-	.406	-
	3						
P-4864		1'-0"		-	-	.27	-
	4						
P-4865		1'-0"		Nil	Nil	.516	Nil
	5						
P-4866		1'-0"		-	-	.605	-
	6						
P-4867		1'-0"		-	-	.69	-
	7						
P-4868		0'-10"		-	-	.772	-
	8						

RED FLAT LATERITE
Ni

Fraction	% Wt.	Lb/Ton	Assay %	Lbs. Ni/Ton	% Total Ni.
Non-Mag.	3.51	70.2	.166	.116	-
Mag.	3.21	64.2	.170	.109	-
2nd. Con.	2.00	40.0	.277	.111	-
3rd. Con.	2.46	49.2	.374	.184	-
Sand	26.87	537.4	.556	2.988	24.2%
Slime	61.95	1239.00	.711	8.819	71.5
Total	100.00	2000.00	.6159% (6163%)	12.327	95.7

RED FLATS LATERITE, CURRY COUNTY

Hole	1	2	3	4	1	2	3	4
	Nickel				Chrome (Composites)			
0								
1	.33	.46	1.17	.357				
2	.362	1.38	.934	.585				
3	.29	.65	.857	.406				
4	.59	.62	1.02	.27		1.38	~~1.23~~ [illegible]	~~3.46~~ [illegible]
5	.695	.62	1.14	.516				
6	.959	1.007	1.129	.605	3.31			
7	1.09	1.29	1.25	.69				
8	1.25			.772				
9	1.34							
10	1.46							
11	1.18							
12								
	Comp.	Comp.	Comp.	Comp.				
	.845	.796	1.04	.516				

RED FLATS LATERITE, CURRY COUNTY

Hole	1	2	3	4	1	2	3	4	
	Nickel				Chrome (Composites)				
0									
1	.33	.46	1.17	.357					
2	.362	1.38	.934	.585					
3	.29	.65	.857	.406					
4	.59	.62	1.02	.27			1.38	~~1.23~~ 3.19	~~3.46~~ 1.53
5	.695	.62	1.14	.516					
6	.959	1.007	1.129	.605		3.31			
7	1.09	1.29	1.25	.69					
8	1.25			.772					
9	1.34								
10	1.46								
11	1.18								
12									
	Comp.	Comp.	Comp.	Comp.					
	.845	.796	1.04	.516					

Sponge Iron on Red Flats Laterite.
1/28/47

Both #1 and #2 in muffle ½ hour max. temp. 1000° to 1200°F.

#1 - Sand tails from conc. test of composite of Holes 1 and 4

Sand tails 70 gms

Charcoal 5 gms

Na_2CO_3 5 gms

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. after 1st mag. separation	37.8	Wt. after 1st separation	33.8
Wt. after washing out charcoal	31.0	Wt. after washing out charcoal and further mag. separation	16.6
Wt. after further separation of mag. material from nonmag.	36.4		

#2 - Slimes 50 gms

Charcoal 5 gms

$CaCO_3$ (P-5614 through 20-mesh) 10 gms

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. 1st mag. separation (contained much carbon)	53.8	Wt. after mag. sep.	9.1
Wt. after 1st washing	52.2		
Wt. after pulv. and 2nd washing	49.3		

Still contains some c

Sponge Iron on Red Flats Laterite.
1/28/47

Both #1 and #2 in muffle ½ hour max. temp. 1000° to 1200°F.

#1 - Sand tails from conc. test of composite of Holes 1 and 4

Sand tails	70 gms
Charcoal	5 gms
Na_2CO_3	5 gms

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. after 1st mag. separation	37.8	Wt. after 1st separation	33.8
Wt. after washing out charcoal	31.0	Wt. after washing out charcoal and further mag. separation	16.6
Wt. after further separation of mag. material from nonmag.	36.4		

#2 - Slimes 50 gms

Charcoal	5 gms
$CaCO_3$ (P-5614 through 20-mesh)	10 gms

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. 1st mag. separation (contained much carbon)	53.8	Wt. after mag. sep.	9.1
Wt. after 1st washing	52.2		
Wt. after pulv. and 2nd washing	49.3		

Still contains some c

RED FLAT SPONGE IRON EXPERIMENT

No. 1 Sands with sodium carbonate and charcoal.

A. Magnetic fraction.

1. Charcoal.

2. Limonite-colored non-opaque - isotropic or weakly birefringent.

3. Small percentage of anisotropic grains.

4. Small percentage of carbonate, effervesces in HCl.

B. Non-magnetic fraction.

1. Charcoal.

2. Carbonate, effervesces briskly in HCl.

3. Magnetic opaque.

4. Limonite-colored non-opaque like that in "3" in "A" above.

No. 2 Slimes with calcium carbonate and charcoal.

A. Magnetic fraction.

1. Charcoal.

2. Limonite-colored but non-opaque material. Either isotopic or very weakly birefringent.

3. Carbonate, effervesces briskly in HCl.

B. Non-magnetic fraction.

1. Charcoal.

2. Carbonate, effervesces in HCl.

3. Unidentified white birefringent material left after HCl treatment.

RED FLAT BIBLIOGRAPHY

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Sponge Iron: Edward H. Robie, Mining and Met. p. 602
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New Use Patterns Required for Survival of Wartime
Metallurgical Innovations: R. S. Dean, Mining and Met.
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RED FLAT BIBLIOGRAPHY

Sponge Iron and Chromium

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New Use Patterns Required for Survival of Wartime Metallurgical Innovations: R. S. Dean, Mining and Met. p. 290, June 1945.

Cle Elum Iron-Nickel Deposits, Kittitas County, Wash. U. S. Dept. of Interior, R. I. 4189, Feb. 1948.

RED FLAT LATERITE

~~TABLE 1~~

11'

DRILL HOLE No. 1

SAMPLE NUMBER	SAMPLE WIDTH	ASSAY Au (OUNCES)	Ag (OUNCES)	Ni (PERCENT)	Hg (PERCENT)
P-4827	1'-2"	NIL	TR	.33	—
-4828	1'-0"	.02	"	.362	NIL
-4829	1'-0"			.29	
4830	1'-0"	NIL	TR	.59	NIL
-4831	1'-0"	TR	"	.695	"
-4833	1'-0"	"	"	.959	"
-4834	1'-0"	NIL	NIL	1.09	"
-4835	1'-0"	.02	TR	1.25	"
-4837	1'-0"	TR	NIL	1.34	"
-4838	1'-0"	NIL	"	1.46	"
P-4839	1'-0"			1.18	

BOTTOM

146
1.29
1.25
.77
4.77
1.19

.33
.46
1.17
.36
2.32
.56

/

RED FLAT LATERITE

~~HOLE 2~~ 7'-3"

DRILL HOLE NO. 2

SAMPLE NUMBER	SAMPLE WIDTH	ASSAY AU (OUNCES)	ASSAY AG (OUNCES)	ASSAY NI (PERCENT)	ASSAY HG (PERCENT)
P-4841	1'-0"	NIL	NIL	.46	–
-4842	1'-0"			1.38	–
-4843	1'-0"	NIL	NIL	.65	NIL
-4844	1'-0"	TR	TR	.62	–
-4845	1'-0"			.62	
-4846	1'-0"	NIL	TR	1.007	NIL
-4847	1'-0"	"	"	1.29	–

0 1 2 3 4 5 6 7 BOTTOM

~~ABD 4848~~

~~4849~~

RED FLAT: LATERITE

~~Hole 3~~ 7'-0"

DRILL HOLE NO. 3

SAMPLE NUMBER		SAMPLE WIDTH	ASSAY Au (OUNCES)	Ag (OUNCES)	Ni (PERCENT)	Hg (PERCENT)
	0					
P-4851	1	1'-0"	NIL	.20	1.17	–
4852	2	1'-0"	"	TR	.934	NIL
4853	3	1'-0"	"	"	.857	"
4854	4	1'-0"			1.02	
4855	5	1'-0"	NIL	TR	1.14	–
4856	6	1'-0"	"	"	1.129	NIL
4857	7	1'-0"	"	"	1.25	–

BOTTOM

8

9

4860

$$\frac{6.48}{1.08}$$

RED FLAT

~~HOLE 4~~ LATERITE 7'-10"

DRILL HOLE NO. 4

SAMPLE NUMBER		SAMPLE WIDTH	ASSAY AU (OUNCES)	ASSAY AG (OUNCES)	ASSAY NI (PERCENT)	ASSAY HG (PERCENT)
	0					
P-4861	1	1-0"	NIL	NIL	.357	NIL
4862	2	1-0"	"	"	.585	"
4863	3	1-0"	-	-	.406	-
4864	4	1'-0"	-	-	.27	-
4865	5	1'-0"	NIL	NIL	.516	NIL
4866	6	1'-0"	-	-	.605	-
4867	7	1'-0"	-	-	.69	-
4868	8	0'-10"	-	-	.772	-

BOTTOM

State Department of Geology and Mineral Industries

702 Woodlark Building
Portland, Oregon

Analyses of Samples Taken by State Department of

Geology from RED FLAT, Curry County, Oregon

1946

Sample number	Description	Assays Au (oz)	Ag (oz)	Ni (%)	Hg (%)
P-4827	Hole no. 1 0'-1'	Nil	Tr	.33	-
P-4828	Hole no. 1 1'-2'	.02	Tr	.362	Nil
P-4829	Hole no. 1 2'-3'			.29	
P-4830	Hole no. 1 3'-4'	Nil	Tr	.59	Nil
P-4831	Hole no. 1 4'-5'	Tr	Tr	.695	Nil
P-4832	Hole no. 1 5'-6'	Tr	Tr	.959	Nil
P-4834	Hole no. 1 6'-7'	Nil	Nil	1.09	Nil
P-4835	Hole no. 1 7'-8'	.02	Tr	1.25	Nil
P-4837	Hole no. 1 8'-9'	Tr	Nil	1.34	Nil
P-4838	Hole no. 1 9'-10'	Nil	Nil	1.46	Nil
P-4839	Hole no. 1 10'-11'	-	-	1.18	-
P-4841	Hole no. 2 0'-1'	Nil	Nil	.46	-
P-4842	Hole no. 2 1'-2'			1.38	-
P-4843	Hole no. 2 2'-3'	Nil	Nil	.65	Nil
P-4844	Hole no. 2 3'-4'	Tr	Tr	.62	-
P-4845	Hole no. 2 4'-5'			.62	
P-4846	Hole no. 2 5'-6'	Nil	Tr	1.007	Nil
P-4847	Hole no. 2 6'-7'	Nil	Tr	.129	-
P-4848	Rocks exposed at Hole no. 2	Nil	Nil	Tr	Nil

702 Woodlark Building
Portland, Oregon

1946 (Cont.)

Sample Number	Description	Au (oz)	Ag (oz)	Ni (%)	Hg (%)
P-4849	Shots from Surface at Hole no. 2	Nil	Nil	.178	
P-4851	Hole no. 3 0'-1'	Nil	.20	1.17	-
P-4852	Hole no. 3 1'-2'	Nil	Tr	.934	Nil
P-4853	Hole no. 3 2'-3'	Nil	Tr	.857	Nil
P-4854	Hole no. 3 3'-4'			1.02	
P-4855	Hole no. 3 4'-5'	Nil	Tr	1.14	-
P-4856	Hole no. 3 5'-6'	Nil	Tr	1.129	Nil
P-4857	Hole no. 3 6'-7'	Nil	Tr	1.25	-
P-4860	Greenstone N.E. of Hole no. 1	Nil	Nil	Nil	
P-4861	Hole no. 4 0'-1'	Nil	Nil	.357	Nil
P-4862	Hole no. 4 1'-2'	Nil	Nil	.585	Nil
P-4863	Hole no. 4 2'-3'	-	-	.406	-
P-4864	Hole no. 4 3'-4'	-	-	.27	-
P-4865	Hole no. 4 4'-5'	Nil	Nil	.516	Nil
P-4866	Hole no. 4 5'-6'	-	-	.605	-
P-4867	Hole no. 4 6'-7'	-	-	.69	-
P-4868	Hole no. 4 7'-8'	-	-	.772	-

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1946 (Cont.)

Sample Number	Description	Au (oz)	Ag (oz)	Cr_2O_3 (%)	Ni (%)
		Assays			
P-5452	Composite of drill hole no. 1	.015		3.53	.845
P-5453	Composite of drill hole no. 2			2.01	.796
P-5454	Composite of drill hole no. 3	.008		3.79	1.04
P-5455	Composite of drill hole no. 4	.004	Tr	1.53	.516
P-5589	Pan Concentrate of drill holes (composite of) 1 & 4 (non-magnetic fraction)			22.77	.166
P-5590	Pan Concentrate of composite of drill holes 1 & 4 (magnetic fraction)			14.58	.170
P-5591	Pan Concentrate of composite of drill holes 1 & 4 2nd concentrate			8.25	.277
P-5592	Pan Concentrate of composite of drill holes 1 & 4 3rd concentrate			5.21	.374
P-5593	Sand Tails composite of drill holes 1 & 4			1.35	.556
P-5594	Slimes Composite of drill holes 1 & 4			1.11	.711

State Department of Geology and Mineral Industries

702 Woodlark Building
Portland, Oregon

Analyses of Samples Taken by State Department of

Geology from RED FLAT, Curry County, Oregon

1947

Sample Number	Description	Assays Cr_2O_3 (%)	Ni (%)
P-6507	Dozer Cut at Drill Hole no. 1 5'-8' Below Collar D Hole no. 1	3.45	0.83
P-6508	Dozer Cut at Drill Hole no. 1 8'-11' Below Collar D Hole no. 1	0.35	0.86
P-6509	Dozer Cut at Drill Hole no. 1 11'-12' Below Collar D Hole no. 1, Soft Serpentine	0.25	0.92
P-6510	Dozer Cut at Drill Hole no. 1 11'-12' Below Collar D Hole no. 1, Mostly Laterite	0.31	0.79
P-6511	Dozer Cut at Drill Hole no. 1 12'-13' Below Collar D Hole no. 1, 70% Laterite 30% Serpentine	0.28	0.74
P-6512	Dozer Cut at Drill Hole no. 1 13'-14' Below Collar D Hole no. 1, 70% Laterite 30% Serpentine	0.43	0.97
P-6513	Dozer Cut at Drill Hole no. 1 14'-15' Below Collar D Hole no. 1, Laterite	0.55	0.98
P-6514	Dozer Cut at Drill Hole no. 1 14'-15' Below Collar D Hole no. 1, Partially Weathered Serpentine	0.32	0.79
P-6515	Dozer Cut East of Walsh Camp (Red Gold) Serpentine from Sump	0.47	0.26
P-6516	Dozer Cut East of Walsh Camp (Red Gold) 0'-2½' Laterite	2.86	1.02
P-6517	Dozer Cut East of Walsh Camp (Red Gold) 2½'-5' Laterite	2.83	0.65
P-6518	Dozer Cut East of Walsh Camp (Red Gold) 5'-6' Laterite	2.12	0.55

State Department of Geology and Mineral Industries

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1947 (Cont.)

Sample Number	Description	Assays Cr_2O_3 (%)	Assays Ni (%)
P-6519	Hedderley Dozer Cut Nearest Pyramid L. O. Grab of Sump	0.47	0.316
P-6520	Hedderley Dozer Cut West of First Cut Serpentine	0.41	0.246
P-6521	Dozer Cut West of Walsh Camp (Red Gold) Garnierite on Crack and Faces	0.74	1.13
P-6522	Dozer Cut West of Walsh Camp (Red Gold) Soft Red Laterite	0.86	1.42
P-6523	Dozer Cut West of Walsh Camp (Red Gold) Bottom of Cut - Serpentine	0.49	0.76

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RED FLAT LATERITE
Ni

Fraction	% Wt.	Lb/Ton	Assay %	Lbs. Ni/Ton	% Total Ni.
Non-Mag.	3.51	70.2	.166	.116	-
Mag.	3.21	64.2	.170	.109	-
2nd. Con.	2.00	40.0	.277	.111	-
3rd. Con.	2.46	49.2	.374	.184	-
Sand	26.87	537.4	.556	2.988	24.2%
Slime	61.95	1239.00	.711	8.819	71.5
Total	100.00	2000.00	.6159% (6163%)	12.327	95.7

RED FLAT SPONGE IRON EXPERIMENT

No. 1 Sands with sodium carbonate and charcoal.

A. Magnetic fraction.

1. Charcoal.
2. Limonite-colored non-opaque - isotropic or weakly birefringent.
3. Small percentage of anisotropic grains.
4. Small percentage of carbonate, effervesces in HCl.

B. Non-magnetic fraction.

1. Charcoal.
2. Carbonate, effervesces briskly in HCl.
3. Magnetic opaque.
4. Limonite-colored non-opaque like that in "3" in "A" above.

No. 2 Slimes with calcium carbonate and charcoal.

A. Magnetic fraction.

1. Charcoal.
2. Limonite-colored but non-opaque material. Either isotropic or very weakly birefringent.
3. Carbonate, effervesces briskly in HCl.

B. Non-magnetic fraction.

1. Charcoal.
2. Carbonate, effervesces in HCl.
3. Unidentified white birefringent material left after HCl treatment.

Sponge Iron on Red Flats Laterite.
1/28/47

Both #1 and #2 in muffle $\frac{1}{2}$ hour max. temp. 1000° to 1200°F.

P-5593 #1 - Sand tails from conc. test of composite of Holes 1 and 4

Sand tails 70 gms

Charcoal 5 gms

Na_2CO_3 5 gms SiO_2 18.36

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. after 1st mag. separation	37.8	Wt. after 1st separation	33.8
Wt. after washing out charcoal	31.0	Wt. after washing out charcoal and further mag. separation	16.6
Wt. after further separation of mag. material from nonmag.	36.4		

#2 - Slimes 50 gms

Charcoal 5 gms

SiO_2 14.42

P-5594 $CaCO_3$ (P-5614 through 20-mesh) 10 gms

Cover of charcoal

Mag. fraction	gms	Nonmag. fraction	gms
Wt. 1st mag. separation (contained much carbon)	53.8	Wt. after mag. sep.	9.1
Wt. after 1st washing	52.2		
Wt. after pulv. and 2nd washing	49.3		

Still contains some c

COMPOSITE OF RED FLAT HOLE 1 & 4 FROM DANNING RESULTS.

NI

WT	%	
28.	.166	104.648 ✓
25.7	.170	.04.369 ✓
16.	.277	.04.432 ✓
19.7	.374	.07.368 ✓
215.	.556	1.19.540 ✓
495.6	.711	3.52.372
800.0	2.254	4.92.729

= .6159 NI

Cr_2O_3

	%	
28.	22.77	6.37.
25.7	14.58	3.74
16.	8.26	1.32
19.7	5.21	1.03
215.0	1.35	2.90
495.6	1.11	5.51
800.0		20.87

= 2.61 % Cr_2O_3

FE

28	21.20	.05.93.60
25.7	43.91	.11.28.49
16	27.45	. 4.39.20
19.7	26.35	. 5.19.09
215.	29.46	.63.33.90
495.6	35.90	1.77.92.04
800.0	184.27	2.68.56.32

= 33.508 % FE

HOLE 1 COMP NI .845 11 FT = 9.295
" 4 " .516 8 FT 4.128 = .706 % NI
19. 13.423

HOLE 1 COMP Cr_2O_3 3.55 11 FT = 3.91
" 4 1.53 8 1.22 = 2.70 % Cr_2O_3
19 5.13

HOLE 1 COMP FE
" 4

STATE OF OREGON DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES
ASSAY LABORATORIES

REQUEST FOR SAMPLE INFORMATION

The State law governing free analysis of samples sent to State Assay Laboratories requires that certain information be furnished the Laboratory regarding samples sent for assay or identification. A copy of the law will be found on the back of this blank. Please fill in the information called for as completely as possible, and submit it along with your sample. Keep a copy of the information on each sample for your own reference.

Your name in full H. F. Hedderly

Post-office address

Are you a citizen of Oregon ____ Date on which sample is sent ____

Name (or names) of owners of the property

Name of claim sample obtained from RED FLAT (NEAR [illegible])

Location of property or source of sample (describe as accurately as possible below):

County ____ Mining district ____

Township 37S Range 13 Section 18 Quarter section ____

How far from passable road

For what minerals or elements do you wish the sample(s) analyzed Au, Ag, Ni, Co

	Channel (length)	Grab	Pipe	Description
Sample No. 1				
Sample No. 2				

IMPORTANT: A vein sample should be taken in an even channel across the vein from wall to wall. Location of sample in the workings, together with the width measured, should be recorded.

(Signed) ____

DO NOT WRITE BELOW THIS LINE - FOR OFFICE USE ONLY - USE OTHER SIDE IF DESIRED

Description

Sample Number	GOLD oz./T.	GOLD Value	SILVER oz./T	SILVER Value	COBALT	NICKEL		
P-7312	Nil		Nil		0.05%	0.21%		

Report issued ____ Card filed ____ Report mailed ____ Called for ____

SIR-5

Red Flats Laterite

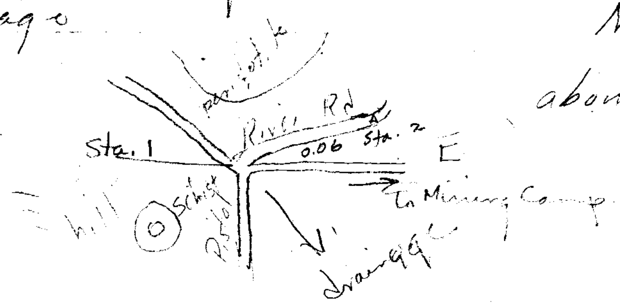
Traverse Notes   Ralph   Lowry

Sta. No. 1 Jct. of 3 Rds.   N. 73°E   N. 72°E   to Sta. No. 2 E on Pistol R. Rd

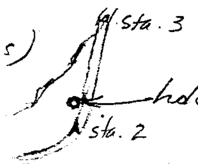
Elev. barometer 2260 ft.   due E   sta. 1 on Mining Camp Rd

0.00 mileage   N 50° W   Rd leading to NW

about due S   Pistol R. Rd to S



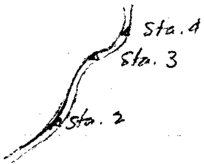
Sta. No. 2 on Pistol R. road. 0.06 mi E. of sta. 1, 5± above sta. 1, approx 2265 ft.   (0° degrees)   N. 15°E.   N. 12°E   to Sta. 3 on Pistol R. Rd



AT HOLE NO 3 (0.095 mileage ~~sta. 3~~)

Sta. No. 3   N. 12°E   N 42°E to sta. 4 on Pistol R. Rd

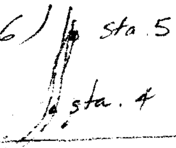
0.15 mileage ~~from Sta. 2~~



3½° above sta. 2 in elevation. (Aneroid 2285 ft.)

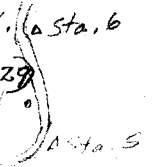
Sta. No. 4 (mileage 0.16)   N. 19°E   N. 15°E.   to sta. 5 on Pistol R. Rd

5 ft. above sta. 3



N. 23° W.   N. 25° W.   to sta. 6 on Pistol R. Road

Sta. No. 5 (mileage 0.27) just E of road. N. 30° W. to milepost 14 (mileage 0.29)



Aneroid 2335 ft.   5° from sta. 5 up to sta. 6

0.35 mileage at spot where parked NE. of hole 2.

RED FLAT

		% Cr_2O_3	% Ni
P-6507	'DOZER CUT #1 5'-8' BELOW COLLAR DH HOLE No. 1	3.45	0.83
6508	" " " 8'-11' " "	0.35	0.86
6509	" " " 11-12' " " SOFT SERPENTINE	0.25	0.92
6510	" " " 11-12 " " MOSTLY LATERITE	0.31	0.79
6511	" " " 12-13' " " 70% LATER. 30% SERP.	0.28	0.74
6512	" " " 13-14 " " " " " "	0.43	0.97
6513	" " " 14-15 " " LATERITE	0.55	0.98
6514	" " " 14-15 " " PARTIALLY WEATHERED SERP	0.32	0.79
6515	'DOZER CUT EAST OF WALSH CAMP - SERP. FROM SUMP	0.47	0.26
6516	" " " " " " 0'-2½' LATERITE	2.86	1.02
6517	" " " " " 2½'-5' "	2.83	0.65
6518	" " " " " 5'-6' "	2.12	0.55
6519	HEDDERLEY 'DOZER CUT NEAREST PYRAMID L.O. GERBOE SUMP	0.47	0.316
6520	" " " WEST OF FIRST CUT. SERP.	0.41	0.246
6521	'DOZER CUT WEST OF WALSH CAMP. GARNIERITE ON FACES CRACKS	0.74	1.13
6522	" " " " " " SOFT RED LATERITE	0.86	[illegible]
6523	" " " " " " Bottom of cut-serp.	0.49	0.76
6524	" " " " " " " " " "		
6525	"BADLANDS" Grab from top of lump	0.52	0.21
6526	" Placer mtl. from Head of Gulch N. side		
6527	Darlingtonia Plants Above flycatcher Sps.		
6528	Buck Brush Leaves " "		

STATE DEPARTMENT OF GEOLOGY
AND MINERAL INDUSTRIES
702 WOODLARK BUILDING
PORTLAND 5, OREGON
December 15, 1946

Sample submitted by R. S. Mason (D.O.G.A.M.I.)

Analysis by:

Sample received on November 18, 1946

L. L. Hoagland
Assayer

Analysis requested Platinum group, Nickel, Chrome assay

Red Flat laterite composite samples

Lab. No.	Sample Marked	Results of Analysis		Remarks
P-5452	Hole #1	Platinum group	Nil	-------------
		Nickel (Ni)	0.845%	
		Chrome (Cr_2O_3)	3.51%	
P-5453	Hole #2	Platinum group	Nil	-------------
		Nickel (Ni)	0.796%	
		Chrome (Cr_2O_3)	1.38%	
P-5454	Hole #3	Platinum group	Nil	-------------
		Nickel (Ni)	1.04%	
		Chrome (Cr_2O_3)	1.23%	
P-5455	Hole #4	Platinum group	Nil	-------------
		Nickel (Ni)	0.516%	
		Chrome (Cr_2O_3)	3.46%	

The Department did not participate in the taking of this sample and assumes responsibility only for the analytical results.

/

General Laboratory Number P 4848 Date received Nov 8 46

Spectrographic Laboratory Number Sample received from Dr Lowry

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Si Fe Mg

2. Elements present in concentrations 10% - 1%.

Chem.

N: trace

3. Elements present in concentrations 1% - 0.1%.

Al Ca

4. Elements present in concentrations 0.1% - .01%.

Na Mn Cr V Cd Ni

5. Elements present in concentrations .01% - .001%.

K Ti Zr Sn Ca Sr Co B

6. Elements present in concentrations below .001%.

Ba — Ag Mo (May be in carbons)

~~Dr. H. C. Harrison~~, Spectroscopist

T C Matthews ....

Peridotite near hole 2

General Laboratory Number P 4827 Date received Nov 8 46

Spectrographic Laboratory Number Sample received from Dr. Lowry

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Fe

2. Elements present in concentrations 10% - 1%.

Si Al

chem.

Ni 0.33

3. Elements present in concentrations 1% - 0.1%.

Mg Cr Cd

4. Elements present in concentrations 0.1% - .01%.

Ca Na Mn Sn V B Ti Ni

5. Elements present in concentrations .01% - .001%.

K Zr Cu Sr Co

6. Elements present in concentrations below .001%.

Mo Ba

Mo not in
P-4848

Trace — Ag

~~Dr. H. G. Harrison~~, Spectroscopist

T C Matthews

0'-1'2" hole no. 1

General Laboratory Number P 4838 Date received Nov 8-46

Spectrographic Laboratory Number Sample received from Dr Lowry

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Si Fe

2. Elements present in concentrations 10% - 1%.

Mg

Ni
Chem. 1.46%

3. Elements present in concentrations 1% - 0.1%.

Al Cr Cd

4. Elements present in concentrations 0.1% - .01%.

Ca Na Mn Sn V Ni

5. Elements present in concentrations .01% - .001%.

K Ti Zr Cu Co B

6. Elements present in concentrations below .001%.

Mo Ba

Mo not present in P-4848

Trace of Ag

~~Dr. H. C. Harrison,~~ Spectroscopist

E. C. Matthews

10' - 11' from hole 1

General Laboratory Number P 4849 Date received Nov 8 46

Spectrographic Laboratory Number Sample received from Dr Lowry

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Al Fe

2. Elements present in concentrations 10% - 1%.

Si

3. Elements present in concentrations 1% - 0.1%. chem. Ni 0.178

?

Mg Na Cr Cd

4. Elements present in concentrations 0.1% - .01%.

Ca Mn Ti V Ni

5. Elements present in concentrations .01% - .001%.

K Zr Sn Mo Sr Cu Co B

6. Elements present in concentrations below .001%.

Ba

Mo not in P - 4848

~~Dr. H. G. Harrison,~~ Spectroscopist

T Chatthews ....

"Shot" from surface
near hole 2

24 Jan 1997

STATE DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES

702 WOODLARK BUILDING
PORTLAND 5, OREGON

Sample submitted by Dr Lowry

Sample received on ____________

Analysis requested Na

Analysis by: [illegible signature]

Lab. No.	Sample Marked	Results of Analysis	Remarks

A special plate was run for the Sodium lines 5889.9 and 5895.9.

Samples in order of Na %:

P-4849 — Highest

P-4848

P-P4838

P-4827 Lowest

The Department did not participate in the taking of this sample and assumes responsibility only for the analytical results.

NOTICE OF PLACER LOCATION

No. 1

Notice is hereby given that the undersigned claiming by right of discovery, location and possession, have associated ourselves together and having compled with the requirements of the mining laws of the United States of America and particularly the provisions of Chapter 2 of Title 30, United States Code Annotated and local customs, laws, and regulations, have located and do hereby this 1st day of July, 1936, locate an Association Placer Mining Claim to be known as "Red Gold Association No. 1 Placer", said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows: Beginning at the section corner at the southeast corner of Section 25 Township 37, South, Range 14 West of the Willamette Meridian, Oregon, running thence East 2640 feet, running thence North 2640 feet, running thence West 2640 feet, running thence South 2640 feet to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the southwest quarter of Section 30, Township 37 South, Range 13 West of the Willamette Meridian.

Discovered: June 28th, 1936

Located: July 1st, 1936

Witness:

H. L. Cooper

L. A. Liljeqvist

Alfred L. LaChance

Carl Smedberg

Mary Smedbert

Harley Gardner
By A. L. La Chance
Attorney in Fact.

Ruby Gardner
By A. L. LaChance
Attorney in Fact
John D. Black
By Carl Smedberg

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows:

Beginning at the section corner at the northeast corner of Section 25, Township 37 South, Range 14 West of the Willamette Meridian, Oregon, running thence East 2640 feet, running thence south 2640 feet, running thence west 2640 feet running thence north 2640 to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the northwest quarter of Section 30, township 37 south, range 13 west of the Willamette Meridian.

Filed and recorded July 1st, 2:30 P.M., 1936 ________________ County Clerk.

Vol. 10 Page 511

-0-

RED GOLD ASSOCIATION NO. 3

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganised), in Curry County, Oregon and described as follows :

Beginning at the section corner at the southeast corner of Section 24, Township 37, South, Range 14 West of the Willamette Meridian, Orégon, running thence East 2640 feet, running thence North 2640 feet, running thence West 2640 feet, running thence South 2640 feet to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the southwest quarter of Section 19, Township 37 South, Range 13 West of the Willamette Meridian.

Filed and recorded July 1st, 2:30 P.M., 1936 ______________ County Clerk.

Vol. 10 Page 512

-0-

RED GOLD ASSOCIATION NO. 4

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganised) in Curry County, Oregon, and described as follows:

Beginning at the section corner at the northeast corner of Section 24, Township 37 South, Range 14 West of the Willamette Meridian, Oregon, running thence East 2640 feet, running thence south 2640 feet, running thence west 2640 feet, running thence north 2640 feet to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the northwest quarter of Section 19, Township 37 South, Range 13 West of the Willamette Meridian.

Filed and recorded July 1st, 2:30 P.M., 1936_______________County Clerk.
Vol. 10 Page 512-13

-0-

RED GOLD ASSOCIATION NO. 5

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows:

Beginning at the section corner at the southeast corner of Section 13, Township 37, South, Range 14 West of the Willamette Meridian, Oregon, running thence East 2640 feet, running thence North 2640 feet, running thence West 2640 feet, running thence South 2640 feet, to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the southwest quarter of Section 18, Township 37 South, Range 13 West of the Willamette Meridian.

Filed and recorded July 1st, 1936. 2:30 P.M.____________County Clerk.
Vol. 10 page 513

-0-

RED GOLD ASSOCIATION NO.6

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows:

Beginning at a wooden stake 4 inches in diameter, 4 feet high abo e the surface of the ground and set in a mound of rock 2 feet high, which stake is set at a point 2640 feet east of the quarter section corner on the east side of section 25, Township 37 South, Range 14 West, of the Willamette Meridian, Oregon, running thence east 2640 feet running thence north 2640 feet running thence west 2640 feet, running thence south 2640 to the place of beginning.

The above claim is surveyed and it is thought that when surveyed this location will equal the northeast quarter of Section 30, Township 37 South, Range 13 West of the Willamette Meridian.

Filed and recorded July 1st, 2:30 P. M. 1936 ___________County Clerk.
Vol. 10 Page 514

-0-

RED GOLD ASSOCIATION NO. 7

Said claim being 160 acres of pâacer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows:

Beginning at a wooden stake 4 inches in diameter, 4 feet high above the surface of the ground and set in a mound of rock 2 feet high, which stake is set at a point 2640 feet east of the quarter section corner on the east side of Section 25, Township 37 South, Range 14 West of the Willamette Meridian Oregon running thence east 2640 feet, running thence south 2640 feet, running thence west 2640 feet, running thence north 2640 feet to the place of beginning.

The above claim is unsurveyed and it is thought that when surveyed this location will equal the southeast quarter of Section 30, Township 37 South, Range 13 West of the Willamette Meridian.

Filed and recorded July 1st, 1936, 2:30 P.M.______________County Clerk.
Vol. 10 Page 514-15

-0-

RED GOLD ASSOCIATION NO. 8

Said claim being 160 acres of placer mining ground situated in the Hunter's Creek Mining District (Unorganized), in Curry County, Oregon, and described as follows:

The southeast quarter of the southeast quarter of section 13, the east half of the northeast quarter of Section 24 and the northeast quarter of the southeast quarter of Section 24, all in township 37 south, range 14 West of the Willamette Meridian, Curry County, Oregon.

Filed and recorded July 1st, 1936, 2:30 P.M.__________________County Clerk.
Vol. 10 page 515

-0-

Glade Creek quicksilver (Red Flats) Gold Beach Area Curry County

The claim owner, M.C. Chapin, volunteered to take me to Red Flats. This aroused a storm of protest from Shannon, McWilliams and the remainder of the Red Flat crowd, who asked Shapin is he wanted to "queer the deposit" by taking me up there. Chapin, however, according to his statements, said that if there was anything "not on the up and up" about the prospect, he wanted to know it.

Chapin has leased the property to Shannon, McWilliams and Sneed for development. The contract is strongly in favor of Chapin and requires a monthly payment by Shannon, regardless of the work or lack of it done. Shannon has been trying to get the contract changed in his favor, but Chapin is holding to the original contract.

The entire proposition has been attacked in a haphazard manner. The testing of the deposit, if done as statements indicate, is the best part of the work, yet no test pitting, no drilling, and no scientific program for sampling has been done. All the samples have been taken from the surface. The road from the surface of Red Flat to Glade Creek has not been surfaced and has been torn up by hauling in wet weather. The water wheel and flume have apparently been built to impress prospective investors, as has the mess house. The engineering work done on the property was stated by S.O. Newhouse to be "lousy" in general.

M.C. Chapin believes that quicksilver is present on the property, yet feels that Shannon claims a far too high value, gives his estimate of 7 to 8 pounds, per ton, of quicksilver. Chapin has become disgusted with Shannon, and on the quiet asked a friend of his, John Collier, to visit the property. Collier is reputed to have had considerable experience with quicksilver deposits, likewise feels that there may be quicksilver on the property, yet is disgusted with the work of Shannon and Sneed.

Shannon has admitted to Chapin that he has failed in two previous ventures, and that this is his last opportunity to make good. Shannon admitted to me that he was no engineer (perhaps his not being a registered engineer might stop the promotion scheme), yet I understand from other sources that his wife is an experienced metallurgist. McWilliams showed me the flow sheet of the plant, which, similar to the lack of planning shown on the property, shows an amazing lack of knowledge of economic operation of a quicksilver plant, involving first a washing of all the ore by hot water, operating in a closed circuit. Further details of the plant are unnecessary.

Malcolm Sneed, associated with Shannon and McWilliams, left for Louisiana early in January, apparently with the intention of not coming back. Several people in Gold Beach and vicinity were left with sizeable bills unpaid, and no promise of payment.

CONFIDENTIAL

State Department of Geology and Mineral Industries

702 Woodlark Building
Portland, Oregon

Apparently, considerable stock has been sold in the Red Flat promotion scheme. The owner of a growery store north of Gold Beach stated that Sneed, who formerly worked with Parker Methods, had offered to pay the grocery bill incurred by Parker Methods, a sum amounting to $80, if the owner of the store would accept $100 in Red Flat stock and pay the difference ($20) in cash. Sneed suggested that the owner of the store get in touch with Shannon, and also gave him the names of a number of stockholders to whom he could write. The store owner volunteered to give me the names after I recommended that he should not invest money in "risky mining stock". These stockholders are:

Seufert Bros., % Ed Seufert, The Dalles, Oregon
McKee, Portland Gas and Coke Co., has stock listed in his wife's name.
Gensler, Portland.
Roy Stevens and R.L. Kenney, % Vancouver Aluminum Plant
Jimmie Grant, Portland Gas and Coke Company
Macwaters, vice-president of Zellerback Paper Company

A number of samples were taken at various places on the Chapin property, and also in the Red Flat group of claims, believed to be on the Al LaChance claim. Samples were taken indiscriminately, but were taken at places designated by Chapin as the locations most favored by Shannon and others interested. The results of these assays, made by the Grants Pass office, are included with this repo rt.

CONFIDENTIAL

It is apparent that the scheme is purely one of promotion. Shannon and McWilliams may have deluded themselves at one time that quicksilver was present. Giving them the benefit of the doubt, if the operation is not purely promotional, it is an example of the rankest incompetence.

The metallurgical processes used and recommended by Shannon and McWilliams have been tested by the Grants Pass Office with negative results in all cases.

Recommendations: Prospecting for nickel and chromite should be encouraged and recommended. No traces of nickel were found, however, yet nickel may occur. Chromite possibilities are by far the best possibilities.

Informants: Lon Shannon, R.I. McWilliams, M.C. Chapin, Rose Chapin, John Collier, S.O. Newhouse, Adrian Schroeder.

R. E. Brown

.776 -

Sample number	Sample width		Assay Au (ounces)	Ag	Ni (percent)	Hg
P-4847	1'-0"	Hole no. 2	Nil	Tr	1.29	-
		Hole no. 3				
P-4851	1'-0"		Nil	.20	1.17	-
P-4852	1-0"		"	Tr	.934	Nil
P-4853	1'-0"		"	"	.857	"
P-4854	1'-0"				1.02	
P-4855	1'-0"		Nil	Tr	1.14	-
P-4856	1'-0"		"	"	1.129	Nil
P-4856	1'-0"		"	"	1.25	-
P-4861	1-0"	Hole no. [illegible]	Nil	Nil	.357	Nil
P-4862	1'-0"		"	"	.585	"
P-4863	1'-0"		-	-	.406	-
P-4864	1'-0"		-	-	.27	-
P-4865	1'-0"		Nil	Nil	.516	Nil
P-4866	1'-0"		-	-	.605	-
P-4867	1'-0"		-	-	.69	-
P-4868	0'-10"		-	-	.772	-

RED FLAT LATERITE

Sample number		Sample width		Assay Au (ounces)	Ag	Ni (percent)	Hg
P-4827	0	1'-2"	Hole no. 1	Nil	Tr	.33	-
P-4828	1	1'-0"		.02	"	.362	Nil
P-4829	2	1'-0"				.29	
P-4830	3	1'-0"		Nil	Tr	.59	Nil
P-4831	4	1'-0"		Tr	Tr	.695	Nil
P-4833	5	1'-0"		Tr	Tr	.959	"
P-4834	6	1'-0"		Nil	Nil	1.09	"
P-4835	7	1'-0"		.02	Tr	1.25	"
P-4837	8	1'-0"		Tr	Nil	1.34	"
P-4838	9	1'-0"		Nil	"	1.46	"
P-4839	10	1'-0"		-	-	1.18	-
	11						
P-4841	0	1'-0"	Hole no. 2	Nil	Nil	.46	-
P-4842	1	1'-0"				1.38	-
P-4843	2	1'-0"		Nil	Nil	.65	Nil
P-4844	3	1'-0"		Tr	Tr	.62	-
P-4845	4	1'-0"				.62	
P-4846	5	1'-0"		Nil	Tr	1.007	Nil

RED FLAT

Samples taken in 1947

		% Cr_2O_3	% Ni
P-6507	Dozer Cut #1 5' - 8' Below Collar D. Hole No. 1	3.45	0.83
6508	" " " 8' - 11' " " " " " "	0.35	0.86
6509	" " " 11' - 12' " " " " " " Soft Serpentine	0.25	0.92
6510	" " " 11' - 12' " " " " " " Mostly Laterite	0.31	0.79
6511	" " " 12' - 13' " " " " " " 70% Laterite, 30% Serpentine	0.28	0.74
6512	" " " 13' - 14' " " " " " " " " " "	0.43	0.97
6513	" " " 14' - 15' " " " " " " Laterite	0.55	0.98
6514	" " " 14' - 15' " " " " " " Partially Weathered Serpentine	0.32	0.79
6515	Dozer Cut East of Walsh Camp - Serpentine from Sump	0.47	0.26
6516	" " " " " " 0' - 2½' Laterite	2.86	1.02
6517	" " " " " " 2½' - 5' "	2.83	0.65
6518	" " " " " " 5' - 6' "	2.12	0.55
6519	Hedderley Dozer Cut Nearest Pyramid L. O. Grab of Sump	0.47	0.316
6520	" " " West of First Cut - Serpentine	0.41	0.246
6521	Dozer Cut West of Walsh Camp - Garnierite on Crack and Faces	0.74	1.13
6522	" " " " " " Soft Red Laterite	0.86	1.42
6523	" " " " " " Bottom of Cut - Serpentine	0.49	0.76

Lab. No.	Description	Assays	
	Woodcock Mountain	Ni	Cr_2O_3
P-6529	Redbird No. 2, Hole 1 0" - 1'	1.45	0.94
P-6530	" " ", Loc cut channel 2' - 3'	1.38	0.94
P-6531	" " ", Hole No. 2 0'6" - 1'6"	1.03	2.65
P-6532	" " ", " " " 1'6" - 2'6"	1.38	2.26
P-6533	" " ", " " " 2'6" - 3'6"	1.24	1.85
P-6534	Grab of inclined shaft dump	0.40	0.34
P-6535	" " top of inclined shaft	1.33	0.94
P-6536	" " unweathered peridotite top of mountain	0.23	0.40
P-6537	Claim No. 4 Loc cut 2½" - 4'0" laterite	0.67	1.50
P-6538	Grab of soil at top of mtn. 6" below surface	0.88	2.12
	Nickel Mountain		
P-6539	Laterite from big cut 2'3" - 4'0"	0.53	1.60
P-6540	" " " " 4'0" - 6'0"	0.66	1.41
P-6541	" " " " 6'0" - 8'0"	0.65	1.11
P-6542	" " " " 8'0" - 2'0"	0.67	0.89
P-6543	Surface of saddle "potential ore" zone	0.60	1.48
P-6544	Top of ridge Hole No. 1 6" - 1'0"	1.72	1.26
P-6545	" " " " " " 1'0" - 2'0"	2.37	0.93
P-6546	" " " " " " 2'0" - 3'0"	2.30	1.06
P-6547	" " " " " " 3'0" - 4'0"	1.65	0.77
P-6548	" " " " " " 4'0" - 5'0"	1.97	0.74

Lab. No.	Description	Assays	
	Nickel Mountain (Cont.)	Ni	Cr_2O_3
P-6549	Top of ridge Hole No. 1 5'0" - 6'0"	2.01	0.77
P-6550	" " " " " " 6'0" - 7'0"	1.85	0.87
P-6551	" " " " " " 7'0" - 8'0"	1.79	0.84

.115 -

Samphe number		Sample width		Assay Au (ounces)	Assay Ag (ounces)	Assay Ni (percent)	Assay Hg (percent)
	6						
P-4847		1'-0"	Hole no. 2	Nil	Tr	1.29	-
	7						
	0		Hole no. 3				
P-4851	0	1'-0"		Nil	.20	1.17	-
	1						
P-4852		1-0"		"	Tr	.934	Nil
	2						
P-4853		1'-0"		"	"	.857	"
	3						
P-4854		1'-0"				1.02	
	4						
P-4855		1'-0"		Nil	Tr	1.14	-
	5						
P-4856		1'-0"		"	"	1.129	Nil
	6						
P-4858		1'-0"		"	"	1.25	-
	7						
P-4861	0	1-0"	Hole no. #	Nil	Nil	.357	Nil
	1						
P-4862		1'-0"		"	"	.585	"
	2						
P-4863		1'-0"		-	-	.406	-
	3						
P-4864		1'-0"		-	-	.27	-
	4						
P-4865		1'-0"		Nil	Nil	.516	Nil
	5						
P-4866		1'-0"		-	-	.605	-
	6						
P-4867		1'-0"		-	-	.69	-
	7						
P-4868		0'-10"		-	-	.772	-
	8						

South end of Ruby Hill Claim
15' of sheer zone composite

Use in Youngberg's report ?

STATE DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES

702 WOODLARK BUILDING
PORTLAND 5, OREGON

General Laboratory Number P 4545 Date received March 27 1946

Spectrographic Laboratory Number 1488 Sample received from F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. **Elements present in concentrations over 10%.**

Silicon, aluminum, iron, magnesium

2. **Elements present in concentrations 10% - 1%.**

3. **Elements present in concentrations 1% - 0.1%.**

Calcium, sodium, manganese, titanium, vanadium

4. **Elements present in concentrations 0.1% - .01%.**

Barium, strontium

5. **Elements present in concentrations .01% - .001%.** 8' - 6'

Chromium, copper 4 - 10

6. **Elements present in concentrations below .001%.**

Nickel

32
60
14) 92 (7

~~Dr. H. C. Harrison, Spectroscopist~~

. . . .

From Nixon - June 11, 194[illegible]

FIRE ASSAY INSTRUCTIONS FOR

RED MOUNTAIN OR RED FLATS ORE

By

R. I. McWillaims

- -

Use as large a crucible as possible on a one ounce per ton basis and proceed as follows:-

Take a one pound composite sample of the ore. Crush and grind it to not less than eighty mesh or finer, and quarter same in the usual manner. Then take a one ounce portion for a one ton assay charge. Use the following fluxing material:-

Sodium carbonate	I, A. T.	ounce.
Litharge	I, A. T.	"
Borax	1/8, A. T.	"
Monteray sand	I, A. T.	"
Flour	1/20, A. T.	"

Mix the above flux with the ore and cover heavily with salt, using about one tablespoonfull or more depending on the size of the crucible, say about 1/4 inch thick or deep, and double the amount of flour.

Put crucible with charge into cold, small furnace with as little draft as possible, and start with a slow fire of about 300° Fh. This allows the mercury to fume and rise and contact the salt which forms a chloride of mercury. At that time increase the fire to the usual fusing heat, keeping the crucible out of a direct draft. Then every fifteen to twenty minutes raise the lid off the furnace and add a mixture of flour and flux to the crucible, and do this at least three times while increasing the heat to the pouring stage. The amount being one tablespoonful of flour and a pinch or two of flux. After pouring into mould, allow to cool in order to get the lead button. Add to the button two milligrams of silver before cupelling. After cupelling, reduce the button with nitric acid solution to purify the gold before weighing.

R. I. McWilliams

From Nixon - June 11, 1941

WET ASSAY
OF
RED MOUNTAIN OR RED FLAT ORE
By
R. I. McWilliams

Directions for grinding and recovery of gold and mercury

Crush to quarter inch mesh, put on bucking board and buck down to 80 mesh or finer. Use one pound of ore for sample. Put same in muller. Add three table spoons full of rock salt. Take one quart of hot water, add eight drops of C. P. sulphuric acid to the quart of hot water. Use one-half of this solution in the muller, grind for fifteen minutes. Then add the other half of solution and grind for an additional fifteen minutes. Wash muller out into gold pan and pan from one pan into another and save all tailings. Let tailings settle and pan very carefully. Then use microscope or glass and you will see the beads of flowered mercury.

FOR A BETTER GOLD RECOVERY, add to this process while grinding, one teaspoonfull of C. P. mercury. Pan down, recovery mercury, put mercury in glass vessel, add two ounces of nitric acid, three ounces of hot water, put same on electric heating plate, keep fire controlled to a medium heat. Must not boil over or too hard. Let same boil until the mercury disappears and then pour off solution carefully and there will be a small button of gold left in the container. Then add a small amount of warm water and pour off again. Set container on slow heat and as soon as same dries, button can then be weighed.

FOR BETTER RESULTS, make a three or four pound grind, using one pound at a time, using the same proportions of solutions and flux. Not necessary to add the new C.P.mercury but use the same mercury in each batch. Weigh the mercury in and weigh it out and the difference is the mercury recovered from the ore.
After all this has been done, save all tailings and grind finer, and again treat for a further recovery of values.
Use one vessel for all tailings and do not pour water off until it is clear as all cloudy water carries values.

IT IS IMPORTANT TO GRIND SLOWLY. ABOUT 30 R.P.M.

I have had excellent results from flotation tests which I have made. My flow sheet for a 100 ton plant has flotation as the final treatment.

R. I. McWilliams.

From Nixon - June 11, 1941

SIMPLE TEST FOR MERCURY

RED MOUNTAIN OR RED FLATS ORE

By

R. I. McWilliams

Use about one pound of ore.

Crush ore to I/8 mesh. DRY.

Grind a small portion (dry) and slowly (about 30 R.P.M.) in hand muller and add about one tablespoonfull of soda ash.

Screen fines through 16 mesh and put oversize back in muller.

Add more of the 1/8 mesh ore and add some soda ash.

Keep doing this until all of sample has been ground to minus 16 mesh, then put all in muller and grind slowly to minus 60 mesh, then add hot, weak solution of sulphuric acid and grind slowly for about twenty minutes.

Solution to consist of 10 or 12 drops of C.P. Sulphuric acid to One quart of hot water.

Then pan carefully taking care not to float off any values.

From Nixon
June 11, 194[illegible]

FLOW SHEET

The flow sheet contemplated by Mr. R. I. McWilliams for handling our Red Mountain or Red Flats ore will consist of the following standard equipment:-

Crusher.

Rolls.

Mill to grind to 100 mesh, having center discharge, no screen

Dore Classifier.

When the pulp leaves the mill the hot water (sulphuric acid) solution should be added in the ratio of three parts of solution to one part of pulp.

From there it goes to a mercury trap, rubber lined, with mechanical feed, also rubber lined. This trap has a compressed air attachment. The air keeping the contents thoroughly agitated in transit. This is a McWilliams invention.

From the mercury trap overflow, the pulp goes to concentration by tables, jigs or other suitable means. The final tailings go to flotation cells.

Curry County - on Wimer Rd.
T41S R10W. 1/4 mi N. of line
between Secs. 12 & 13

THIN LATERITE 1FT. IN
GABBRO, SHOWING PINK
MINERAL, FEW BLACK
STREAKS.

General Laboratory Number P-4058 | Date received September 28th 1945

Spectrographic Laboratory Number 1350 | Sample received from F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Silicon, aluminum

2. Elements present in concentrations 10% - 1%.

Iron, titanium, sulphur

Pink from titanium?

3. Elements present in concentrations 1% - 0.1%.

Magnesium

4. Elements present in concentrations 0.1% - .01%.

Sodium, manganese, zirconium, vanadium

5. Elements present in concentrations .01% - .001%.

Calcium

6. Elements present in concentrations below .001%.

Copper, barium, strontium

~~Dr. H. G. Harrison, Spectroscopist~~

E. W. Miller

D-H 67-02

Property: Red Flat - Red Gold Placer #1
Location: NE$\frac{1}{2}$ SW$\frac{1}{4}$ sec. 30 T. 37S., R. 13 W.
Date: June 17, 1967.

Depth % Ni

0-
1.10
-
1.39
10-
0.66
-
0.55
20-
0.44
-

Bottom
at 25 ft.

DH-67-01

Property: Red Flat - Red Gold Placer #2
Location: SW$\frac{1}{4}$ NW$\frac{1}{4}$ sec. 30 T. 37S., R. 13 W
Date: June 17, 1967

Depth %NI

0-
1.05
-
1.24
10-
0.69
-
0.58
20-
0.35
-

Bottom at
25 ft.

c-782

DH-67-03

Property: Red Flat - Red Gold Placer #1
Location: NE$\frac{1}{4}$ SW$\frac{1}{4}$ Sec. 30, T. 37S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.58
-	
	0.94
10-	
	0.47
-	
	0.29
20-	

Bottom at 20 ft.

DH-67-05
Property: Red Flat - Red Gold Placer #2
Location: SE$\frac{1}{4}$ NW$\frac{1}{4}$ sec. 30, T. 37 S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.43
-	
	0.79
10-	
	0.49
-	

Bottom at 15 ft.

DH 67-06

Property: Red Flat - Red Gold Placer #7
Location: NE$\frac{1}{4}$ SE$\frac{1}{4}$ Sec. 30,T.37 S., R. 13 W.
Date: June 19, 1967

Depth	% Ni
0-	
	0.43
-	
	0.74
10-	
	0.49
-	
	0.53
20-	
	0.25
-	

Bottom at 25 ft.

DH-67-04
Property: Red Flat - Red Gold Placer #2
Location: NW$\frac{1}{4}$ NW$\frac{1}{4}$ sec. 30, T. 37 S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.94
-	
	0.36
10-	
	0.31
-	
	0.32
20-	
	0.38
-	

Bottom at 25 ft.

D-H 67-02

Property: Red Flat - Red Gold Placer #1
Location: NE$\frac{1}{2}$ SW$\frac{1}{4}$ sec. 30 T. 37S., R. 13 W.
Date: June 17, 1967.

	Depth	% Ni
0-		
		1.10
-		
		1.39
10-		
		0.66
-		
		0.55
20-		
		0.44
-	Bottom at 25 ft.	

DH-67-01

Property: Red Flat - Red Gold Placer #2
Location: SW$\frac{1}{4}$ NW$\frac{1}{4}$ sec. 30 T. 37S., R. 13 W
Date: June 17, 1967

	Depth	%Ni
0-		
		1.05
-		
		1.24
10-		
		0.69
-		
		0.58
20-		
		0.35
-	Bottom at 25 ft.	

DH-67-03

Property: Red Flat - Red Gold Placer #1
Location: NE¼ SW¼ Sec. 30, T. 37S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.58
-	
	0.94
10-	
	0.47
-	
	0.29
20-	

Bottom at 20 ft.

DH 67-06

Property: Red Flat - Red Gold Placer #7
Location: NE¼ SE¼ Sec. 30,T.37 S., R. 13 W.
Date: June 19, 1967

Depth	% Ni
0-	
	0.43
-	
	0.74
10-	
	0.49
-	
	0.53
20-	
	0.25
-	

Bottom at 25 ft.

DH-67-05

Property: Red Flat - Red Gold Placer #2
Location: SE¼ NW¼ sec. 30, T. 37 S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.43
-	
	0.79
10-	
	0.49
-	

Bottom at 15 ft.

DH-67-04

Property: Red Flat - Red Gold Placer #2
Location: NW¼ NW¼ sec. 30, T. 37 S., R. 13 W.
Date: June 18, 1967

Depth	% Ni
0-	
	0.94
-	
	0.36
10-	
	0.31
-	
	0.32
20-	
	0.38
-	

Bottom at 25 ft.

Gust F. Anderson #1 P-4601

T.37S, R.12 W., sec.2

Curry County

STATE DEPARTMENT OF GEOLOGY
AND MINERAL INDUSTRIES

702 WOODLARK BUILDING
PORTLAND 5, OREGON

General Laboratory Number P 4601 Date received April 26 1946

Spectrographic Laboratory Number 1508 Sample received from F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. **Elements present in concentrations over 10%.**

Iron, arsenic

2. **Elements present in concentrations 10% - 1%.** prob. 1.5% Ni ?

Cu = 5.3%
Au 0.04 oz
Ag 0.70

Nickel, copper, magnesium

3. **Elements present in concentrations 1% - 0.1%.**

Silicon, calcium, manganese, chromium

4. **Elements present in concentrations 0.1% - .01%.**

Aluminum, titanium, zinc, cadmium, cobalt, antimony

5. **Elements present in concentrations .01% - .001%.**

Vanadium, silver

6. **Elements present in concentrations below .001%.**

Bead was analyzed
but no platinum group
elements were present.

~~Dr. H. C. Harrison, Spectroscopist~~

E. H. Mills

Gust F. Anderson #2 P-4602

T-37S, R.12 W., sec.2

Curry County

STATE DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES

702 WOODLARK BUILDING
PORTLAND 5, OREGON

General Laboratory Number P 4602 **Date received** April 26 1946

Spectrographic Laboratory Number 1510 **Sample received from** F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. **Elements present in concentrations over 10%.**

Cu = 30.7 %
Au = 0.1303
Ag = 2.0 "

Iron, copper, arsenic

2. **Elements present in concentrations 10% - 1%.**

Magnesium

3. **Elements present in concentrations 1% - 0.1%.** Ni prob 0.75%?

Silicon, calcium, manganese, chromium, cobalt, nickel

4. **Elements present in concentrations 0.1% - .01%.**

Aluminum, titanium, zinc, cadmium, antimony

5. **Elements present in concentrations .01% - .001%.**

Vanadium, silver

6. **Elements present in concentrations below .001%.**

Bead was analyzed but no platinum or elements of the platinum group were present

~~Dr. H. C. Harrison, Spectroscopist~~

E. H. Miller

THIN LATERITE, 1FT. IN GABBRO, SHOWING PINK MINERAL, FEW BLACK STREAKS.

General Laboratory Number P-4058 Date received September 28th 1945

Spectrographic Laboratory Number 1350 Sample received from F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Silicon, aluminum

2. Elements present in concentrations 10% - 1%.

Iron, titanium, sulphur

3. Elements present in concentrations 1% - 0.1%.

Magnesium

4. Elements present in concentrations 0.1% - .01%.

Sodium, manganese, zirconium, vanadium

5. Elements present in concentrations .01% - .001%.

Calcium

6. Elements present in concentrations below .001%.

Copper, barium, strontium

~~Dr. H. G. Harrison, Spectroscopist~~

E. W. Miller..............

Curry Co.
TWP. 41S. 10W. Sec. 12 + 13.
1/4 mi N of line between Sec. 12 + 13.
Analysis represents black coating of rocks of sample -

1 ft. laterite in gabbro - containing pinle mineral

General Laboratory Number P 4061

Date received October 22 1945

Spectrographic Laboratory Number 1373

Sample received from W. D. Lowry

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Silicon, aluminum, iron, magnesium

2. Elements present in concentrations 10% - 1%.

Calcium, manganese

3. Elements present in concentrations 1% - 0.1%.

Cobalt, nickel

4. Elements present in concentrations 0.1% - .01%.

Sodium, titanium, tin, chromium

5. Elements present in concentrations .01% - .001%.

Vanadium, copper

6. Elements present in concentrations below .001%.

~~Dr. H. C. Harrison, Spectroscopist~~

E. H. Miller
. . . .

Red Flats Laterite - Curry County

STATE DEPARTMENT OF GEOLOGY AND MINERAL INDUSTRIES

702 WOODLARK BUILDING
PORTLAND 5, OREGON

Sample submitted by L. L. Hoagland & F. W. Libbey

Sample received on July 12, 1946

Analysis requested Nickel

Analysis by:

E. W. Miller

Lab. No.	Sample Marked	Results of Analysis	Remarks
P-4827	1591	Nickel - 0.1 % -	Hole #1 - sample #1
P-4830	1592	Nickel - 1.0 - 0.1% .3	Hole #1 - sample #4
P-4834	1593	Nickel - 1.0 - 0.1% .5	Hole #1 -- sample #7
P-4838	1594	Nickel - 1.0 - 0.1% .7	Hole #1 - sample #10
P-4841	1595	Nickel - 0.1 %	Hole #2 - sample #13
P-4844	1596	Nickel - 1.0 - 0.1% .3	Hole #2 - sample #16 3.0-4.0'
P-4847	1597	Nickel 1.0 - 0.1% .9	Hole #2 - sample #16

The Department did not participate in the taking of this sample and assumes responsibility only for the analytical results.

Sample
Carbonaceous
Schist
Near Red Flat
Sent by J A Walsh, Coos Bay

STATE DEPARTMENT OF GEOLOGY
AND MINERAL INDUSTRIES
1069 STATE OFFICE BUILDING
PORTLAND 1, OREGON

General Laboratory Number P-16 094   Date Feb 26 1954

Spectrographic Laboratory Number ______   Sample received from F. W. Libbey

QUALITATIVE SPECTROGRAPHIC ANALYSIS
(Quantities estimated to nearest power of ten)

1. Elements present in concentrations over 10%.

Si

2. Elements present in concentrations 10% - 1%.

Al Fe Mg Ca Na

3. Elements present in concentrations 1% - 0.1%.

K Mn Ti

4. Elements present in concentrations 0.1% - .01%.

Bi Pb V Cu Ba Ni

5. Elements present in concentrations .01% - .001%.

Cr Mo Ag Sr

6. Elements present in concentrations below .001%.

Thomas C. Matthews, Spectroscopist

TCM

CRIB MINERAL RESOURCES FILE 12

RECORD IDENTIFICATION
RECORD NO............ M055858
RECORD TYPE.......... XIM
COUNTRY/ORGANIZATION. USGS
MAP CODE NO. OF REC..

REPORTER
NAME........................... PETERSON, JOCELYN A.
DATE........................... 76 08
UPDATED........................ 81 03
BY............................. FERNS, MARK L. (BROOKS, HOWARD C.)

NAME AND LOCATION
DEPOSIT NAME................... RED FLAT LATERITE
SYNONYM NAME................... RED FLAT PLACER

MINING DISTRICT/AREA/SUBDIST. RED FLAT

COUNTRY CODE................... US
COUNTRY NAME: UNITED STATES

STATE CODE................ OR
STATE NAME: OREGON

COUNTY.................... CURRY
DRAINAGE AREA............. 17100312 PACIFIC NORTHWEST
PHYSIOGRAPHIC PROV........ 13 COAST RANGE

QUAD SCALE	QUAD NO OR NAME
1: 62500	GOLD BEACH

LATITUDE	LONGITUDE
42-20-30N	124-17-23W

UTM NORTHING	UTM EASTING	UTM ZONE NO
4688300.	393750.	+10

TWP...... 037S; 037S
RANGE.... 013W; 014W
SECTION.. 13 24 25; 18 19 30
MERIDIAN. WILLAMETTE

POSITION FROM NEAREST PROMINENT LOCALITY: 17 MI. E. OF GOLD BEACH

LOCATION COMMENTS: UTM COORDINATES ARE FOR CENTRAL PART OF RED FLAT IN SEC. 30, T. 037S., R. 014 W.

COMMODITY INFORMATION
COMMODITIES PRESENT.......... NI CR CO HG AU PT

POTENTIAL.......
OCCURRENCE...... CR CO HG AU PT

ORE MATERIALS (MINERALS,ROCKS,ETC.):
CINNABAR, NATIVE MERCURY

MAIN ORE MINERALS:
CINNABAR, NATIVE MERCURY

COMMODITY COMMENTS:
EARLY OWNERS CLAIMED TO HAVE FOUND AU, AG, PT AND HGLOCALLY. PLACER MINING FOR THESE METALS WAS CONDUCTED IN TH
1930'S AND LATER BUT THERE IS NO RECORD OF PRODUCTION

ANALYTICAL DATA(GENERAL)
THE AVERAGE GRADE OF SOIL AND SAPROLITE IS ABOUT 0.80 % NI, 0.15 % CO, 1.14 % CR2O3, AND 18 % FE

EXPLORATION AND DEVELOPMENT
STATUS OF EXPLOR. OR DEV. 2
PROPERTY IS INACTIVE
PRESENT/LAST OWNER....... HANNA MINING CO., RED FLATS NICKEL CORP., BIG BASIN NICKEL CORP., 1977

DESCRIPTION OF DEPOSIT

DEPOSIT TYPES:
LATERITE
FORM/SHAPE OF DEPOSIT:

SIZE/DIRECTIONAL DATA
SIZE OF DEPOSIT...... MEDIUM
COMMENTS(DESCRIPTION OF DEPOSIT):
THE TOTAL AREA OF LATERITIC SOIL IS ABOUT 1100 ACRES; THE AVERAGE DEPTH OF SOIL AND SAPROLITE IS ESTIMATED TO B
ABOUT 8 FT

DESCRIPTION OF WORKINGS
SURFACE

COMMENTS(DESCRIP. OF WORKINGS):
THE CLAIMS HAVE BEEN EXPLORED BY ABOUT 5 MILES OF PROSPECT ROADS AND 15000 FT OF DOZER TRENCHING AND MANY CHURN
DRILL AND HAND-AUGER HOLES

PRODUCTION
NO PRODUCTION

RESERVES AND POTENTIAL RESOURCES

ITEM ACC AMOUNT THOUS.UNITS YEAR GRADE OR USE

GEOLOGY AND MINERALOGY

AGE OF HOST ROCKS............. JUR
HOST ROCK TYPES............... LATERITE AND SAPROLITE DERIVED FROM PARTLY SERPENTINIZED HARZBURGITE

GEOLOGICAL DESCRIPTIVE NOTES. LARGE, RELATIVELY FLAT, UPLAND AREAS UNDERLAIN BY WEATHERED SERPENTINE AND LATERIT

LOCAL GEOLOGY

GEOLOGICAL PROCESSES OF CONCENTRATION OR ENRICHMENT:
ERDSION

GENERAL COMMENTS
JOHNSON RECORD (M061008) AND COURY RECORD (W000689) MERGED WITH THIS RECORD AND DELETED FROM OREGON FILE.

GENERAL REFERENCES

1) OREGON MINE AND MINERALS HANDBOOK, 1940, V 14C, V1
2) BROOKS, H. C., 1963, QUICKSILVER IN OREGON: OREGON DEPT. OF GEOLOGY AND MINERAL INDUSTRIES, BULL. 55, 223 P.
3) MERCURY IN OREGON, 1965, USBM IC 8252
4) HUNDHAUSEN, R.J., MC WILLIAMS, J.R., AND BANNING, L.H., 1954, PRELIMINARY INVESTIGATION OF THE RED FLATS NICKE DEPOSITS, CURRY COUNTY, ORE., USBM RI 5072 NICKEL-BEARING LATERITE AREAS OF SOUTHWESTERN OREGON: | 5) DOLE, H. AND OTHERS, 1948 , DOGMI "THE ORE BIN", VOL. 10 , NO. 5 , P. 33 - 38 .
6) RAMP, L., 1978, INVESTIGATIONS OF NICKEL IN OREGON: DOGMI MISCELLANEOUS PAPER NO. 20, P. 15-18
7) RAMP, L., 1977, GEOLOGY, MINERAL RESOURCES AND ROCK MATERIALS OF CURRY COUNTY, OREGON: DOGMI BULL. 93, P. 4

Big Basin Group

Red Flats Nickel Corp

Thomas B. Swanton - Rte 2 - Box 220
Phone = 896-3674 Springfield Oregon 97477
Dennis A Winn - P.O. Box 390 Gold Beach, Ore.
(#) Dean J. Brewington 2756 N. Mountain Ave. Claremont, Calif. 91711
L. M. Honaker, 2775 N. Towne Ave. Claremont. Cal. 91711
George E. Motte P.O. Box 2185 Harbor
→ Bill Toner. (Vice Pres) Box 2185 Harbor, Oregon
(503-469-2007)

Wells head of USBM Albany.
Siemens

VESCO ~~Vancouver~~ option on property in
Del Norte.

Glen Colebank

Leon M. Dippold

Stanley "
Clifford "
W. B. Smith
Lester Bradley
Henry C. Shields
Leona Shields
6
18 Verle Godfrey
8
14 Lester W. Barclay

14
2½
1½

RED FLATS LATERITE, CURRY COUNTY

Hole	1	2	3	4	1	2	3	4
	Nickel				Chrome			
					(Composites)			
0								
1	.33	.46	1.17	.357				
2	.362	1.38	.934	.585				
3	.29	.65	.857	.406				
4	.59	.62	1.02	.27		1.38	~~1.23~~ 3.79	~~3.46~~ 1.53
5	.695	.62	1.14	.516				
6	.959	1.007	1.129	.605	3.31			
7	1.09	1.29	1.25	.69				
8	1.25			.772				
9	1.34							
10	1.46							
11	1.18							
12								
	Comp.	Comp.	Comp.	Comp.				
	.845	.796	1.04	.516				

RED FLAT LATERITE
Ni

Fraction	% Wt.	Lb/Ton	Assay %	Lbs. Ni/Ton	% Total Ni.
Non-Mag.	3.51	70.2	.166	.116	-
Mag.	3.21	64.2	.170	.109	-
2nd. Con.	2.00	40.0	.277	.111	-
3rd. Con.	2.46	49.2	.374	.184	-
Sand	26.87	537.4	.556	2.988	24.2%
Slime	61.95	1239.00	.711	8.819	71.5
Total	100.00	2000.00	.6159% (6163%)	12.327	95.7