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April 6, 1977

Nu-Beth
Sundance Project
Core Studies

Preliminary Evaluation of Geochemical Data on Core Samples from Hole SP758R

Introduction:

A preliminary geochemical investigation of the distribution of minerals and elements is in process for core hole #SP758R, on the Sundance Project of the Nu-Beth Joint Venture. The object of the investigation is to determine the relationships of chemical and mineral species to the geochemistry of uranium mineralization. The purpose is three-fold:

1. To provide background data for environmental purposes, i.e., to alert of any potential pollutants contained in the formation which might become mobilized as a result of planned solution mining operations.
2. To provide background data on chemical composition of mineralized and non-mineralized sand which will be essential in planning the mode of in-situ leaching operations and will assist in monitoring the progress of leaching underground during ISL operations.
3. To discover any geochemical relationships which might be helpful as guides in exploration and for ore control in development drilling.

In conjunction with the core studies referred to in this report, leaching studies have also been conducted which may be evaluated in light of these core studies when they are completed. Water quality studies have also been completed and will be evaluated in connection with these core studies when they are completed.

This interim report is to bring up to date the progress which has been made on the core studies to this point. These preliminary core studies are designed to provide an overall view of the geochemical relationships which exist within this core hole and to get a general look at the distribution of all minerals and trace elements which might be related to uranium mineralization. The

results of these studies will set the more specific direction for future studies of core and cuttings samples. Although these initial studies may seem expensive and parts of them wasteful, they are necessary and will pay off in the long run by providing the most effective and efficient techniques for geochemical investigation to be used in more extensive future work on this property.

Lithology:

The core samples provided for study from this hole were dry. A lith-log was prepared, but such things as color may have changed since the fresh core was extracted. The lith-log is attached. The composition of the core ranged from fine to medium-grained sand to very fine-grained silt-stone. There are lenses of clay and very minor calcite throughout the core. The general mineralogy of the core is typical of many of the Wyoming uranium deposits. This deposit, however, is characterized by a dearth of calcite and sulfides. These two components are generally essential to the make-up of a geochemical system conducive to the formation of uranium ore deposits in sandstone. In this area, however, the ground water contains sufficient carbonate to support uranium in solution, the dominant cation being sodium rather than calcium. The reducing capacity needed to cause uranium precipitation to form the ore body is probably provided by the unusual abundance of organic material. These visual observations noted on the lith-log are confirmed by laboratory investigation. The attached certificate of analysis shows the average Ca content to be $<1\%$ in this hole. Likewise, total sulfurs average $<0.1\%$. Heavy liquid separations are dominantly pyrite but also total $<0.1\%$. Carbon estimates are somewhat lower than the 1% visual.

In general, the composition of the sand is $\sim 60\%$ quartz, 35% feldspar, (split $\sim 50\%$ plagioclase + 50% orthoclase), 5% clay, (probably sodium montmorillonite), and minor amounts of calcite, pyrite or marcasite, and carbonaceous material. Limonite is prominent in the oxidized zone by virtue of its color, however, the color can result from a very small amount of pyrite. Other minerals of lesser significance also present in the sand include: magnetite, barite, zircon, rose and smokey varieties of quartz, sericite, muscovite, sphene, almandine, garnet, chlorite and biotite.

Evidence of oxidation is clearly visible due to limonite staining over the interval 413-415 feet.

The lith-log correlates well with the electric log over the cored interval. Visible oxidation just overlies the silt stone at 417'. Unfortunately, the core material provided starts at 406' and does not include mineralization shown on the E-log at 382'. The silt stone at 417-421 feet is substantial and might form an effective aquaclude

if it is continuous over the area. This log and core are typical of the uranium deposits of this area. There are many facies changes over the cored interval, which is typical of sand-stone-type uranium deposits in Wyoming and indicative of a marginal marine depositional environment. Accordingly, most of the clay and calcite lenses intercepted by this hole are not continuous. Correlation of the logs from several holes is necessary to determine whether or not any given clay or silt-stone interval is continuous over the area. Generally, the clay lenses do not affect the permeability of the total formation appreciably; however, they do tend to increase the ratio of horizontal to vertical permeability which is advantageous in helping to control the flow of solution underground. On the basis of the data from this hole alone, it is possible that the oxidation mentioned above is related to the mineralization at 382 feet rather than that shown at 420 feet. More data is necessary to investigate this possibility. It is recommended that future core holes include this upper mineralization in the cored interval.

The mode of occurrence of the uranium is not obvious from visual examination of the core. This information is necessary to determine how much of the uranium is available for beneficiary and how much is tied up in refractory minerals such as zircon, apatite, monazite, etc. It is also important to know the amount of sulfides contained in the mineralization because of their effect on reagent consumption during in-situ-leaching operations.

To get this information, heavy liquid separations were made. The refractory minerals and pyrite become concentrated in the heavy fraction. A copy of these results is attached. In general, the heavy mineral fractions were dominated by pyrite with the combined concentrations <0.1%. This is very low compared with other deposits of this type. The implication is that most of the uranium is available for beneficiation and the effect of sulfides on reagent consumption should be minimal.

The only anomalous concentrations of heavy minerals occur at 410-414 feet and 449-455 feet, above and below the major uranium mineralization; however, any attempt to generalize here with respect to this correlation would be dangerous because of the very low concentrations of heavy minerals and because of the possibility that the concentration at 410-414 feet may not be related to the major uranium mineralization at 417-422 feet.

The screen tests correspond to the lith-log and E-log quite well. There appears to be a slight inverse correlation of grade of U_3O_8 with grainsize in the interval from 420-450 feet. This effect could be mirrored somewhat by permeability, but there are not a sufficient number of permeability samples over the interval to determine whether or not this is in fact the case. In any event, there is a potential problem, but the magnitude of the problem does not appear to be great. It is recommended that these tests be repeated on subsequent cores and that more samples be submitted for permeability testing.

There is an inverse correlation of both total carbon and carbonate with grade of U_3O_8 . This correlates with the observed increase of silty and clay material above and below the major uranium mineralization at 430 feet. These conditions are favorable with respect to reagent consumption, provided solutions are confined to the mineralized zone.

Geochemistry:

The primary trace elements usually associated with uranium are Mo, Se, and V. The distribution of these elements and U is definitive of the direction of the chemical potential which gives rise to the traditional roll fronts and is indicative of the extent, direction of movement, and strength of oxidizing solutions active in forming the roll front.

Uranium is generally in the form of uraninite, coffinite or pitchblende, vanadium as vanadenite or carnotite; molybdenum as molybdenite; and selenium as ferro-selenite. These minerals are stable under reducing conditions. When oxidized they break down and liberate their respective cations. These trace elements can be arranged in order of increasing stability which is also the order in which they would be mobilized under oxidizing conditions: $Mo < (U,V) < Se$. When these ions are in solution and encounter a reducing environment which causes them to precipitate, they will be distributed in a chromatographic sequence just the reverse of their stability. These relationships are especially useful where the oxidation-reduction relationships are not easily detected visually.

The distribution of U, V, Se and Mo shown on the histogram displays the classic relationships which are predicted by thermodynamics. The presence of Mo and Se are mutually exclusive. There is a significant enrichment in Se at the 415 foot level where oxidation is observed in the core. Mo by the same token is depleted in the same region. Unfortunately no core was provided above 406 feet; it would have been interesting to see if the same relationship occurred below the upper mineralization which would indicate that the two mineralized intervals are related and are part of the same geochemical system. It is recommended that on future holes corings begin above the uppermost mineralized intercept. In any case the relationship of Mo and Se is in the classic sense and indicates that the geochemical system is simple rather than complex; that is, that there is probably only one episode of mineralization. Under these circumstances this relationship should persist throughout the system and should facilitate tracing the redox interface. This evidence should be sufficient justification for resampling all drill cuttings from the project and analyzing them for Mo and Se.

The Mo halo on the reduced side of the redox interface tends to be much more wide-spread than the Se on the oxidized side which tends to become enriched at the redox interface. The Mo then should be the key to broadening the exploration target and the Se should be useful in defining clearly the location of the redox interface.

Vanadium is concentrated curiously at the base of the redox interface. There is also a vague direct correlation of V and U. One might also argue that there is a correlation of Mo and U, but it is much less distinct. The enrichment of the V at the base of the redox interface is somewhat difficult to explain, other than to say that it is commonly associated with U. Again, if core were available from the upper mineralization a relationship might be established which would indicate whether or not enrichment of V at this point is due to some influence from the overlying formation.

There also appears to be a very slight correlation of Cu with V. This enrichment in Cu could be due to the formation of chalcopyrite at the redox interface. In areas where there is Cu, its relationship to the sulfides can be a useful tool in exploration.

Phosphorus correlates vaguely with U and could present a slight problem if mobilized during leaching. The phosphates of U are not soluble in carbonate solutions and any P mobilized during leaching will remove U from solution irreversibly. It is not likely that this will present a great problem because of the low solubility of phosphates in carbonate solutions. For this reason Ph control during in-situ leach operations will be important. Fortunately, the low sulfide content will reduce the tendency of the Ph of the leaching solutions to drift downward. Phosphates are soluble in acid solutions. This might be a contributing factor in the reduced efficiency of acid leaching in the agitation leach tests reported by Rosswog.

Calcium, iron and magnesium all correlate well with each other and with carbonate and total carbon. Although present in relatively low abundances, this is an indication that Mg and Ca are probably tied up dominantly in calcite with relatively little Ca in the clays. This is an indication that the clays are dominantly Na-montmorillonites and, therefore, fully expanded. Mobilization of Ca or Mg during ISL operations should not present any problems as they should exchange for Na in the clays which will have a shrinking effect on the clays. Iron is obviously not all tied up in pyrite, in fact the correlation with heavy minerals is not at all apparent. The correlation of Fe with carbonate is entirely consistent with the stability of siderite (iron carbonate) under leaching conditions.

Potassium and sodium correlate only vaguely with each other and with U mineralization. There is some slight correlation of Na with Ca, Mg, and Fe. Potassium and sodium are tied up primarily in feldspars. The depletions shown on the histograms at the redox interface are probably the result of feldspars being altered to clay in an area which is chemically active. Monitoring of these cations

during leaching will be important to monitor the alteration of feldspars. This is important because when feldspars are altered, SiO_2 is released and can cause permeability problems by the formation of silica gels or can poison ion exchange resin in the extraction plant.

There is a slight enrichment in Al in the intervals shown as silty or clay rich on the lith-log. These zones may prove helpful in containing leaching solutions during ISL operations.

There is not a great deal of variation in Si content which is consistent with the observation that the sand is clean and relatively free of clay and other materials. This is also reflected in the relatively high percentages of SiO_2 compared with Al_2O_3 . In most cases these two components account for approximately 90% of the material in the formation which is consistent with observations made on the lith-log. Such conditions would be ideal for leaching both from the operational standpoint and from an environmental standpoint.

Curiously enough, total sulfur content does not correlate well with heavy mineral distribution despite the fact that the heavy mineral separates were dominantly pyrite. This is a reflection of the very low concentration of pyrite and other heavy minerals in the formation. The sulfur does, however, correlate well with uranium mineralization indicating that there is sulfur present in forms other than pyrite. This is consistent with the classic geochemical model of roll fronts that the sulfides are generally present and contribute to, or are a product of, the reducing environment necessary to maintain a uranium ore body. Fluorine does not appear to correlate well with anything consistently except Na. Although F is a common mineralizer in both groundwater and hydrothermal solutions, forming complexes with a variety of transition metal cations, its distribution in this instance does not seem to follow any useful, predictable pattern. Perhaps because of its low concentration.

Chlorine, like F is uniformly low and doesn't conform to any particular systematic pattern. The implication is that Cl build up during ISL operations should come primarily from the extraction plant rather than being mobilized from the formation. This makes Cl much easier to keep track of from a metallurgical accounting standpoint and reduces the potential for Cl contamination of the environment once all the Cl which has been added to the system has been accounted for.

Sulfate is uniformly low in keeping with the rest of the data which is indicative of a strongly reducing environment. Arsenic is another element commonly associated with uranium; however, spot samples taken throughout the hole show a very low level of As. This is favorable environmentally but will not help in exploration or monitoring of ISL Operations.

Nitrate values are uniformly low. This is to be expected. These samples were run primarily for environmental purposes. The implication is that any anomalous NO_3^- measurements made subsequent to leaching would be the result of ISL operations. The oxygen-carbonate system which has been selected should not result in the formation of nitrates unless a nitric acid stripping system is used for elution. Some companies have suggested such a system to gain additional oxidizing capacity from NO_3^- spilled over into the injection solution and this is probably what prompted the states to require that NO_3^- be monitored.

A combined composite sample of mineralized core material from holes SP-758R and SP-829R was prepared for use in laboratory leaching studies. This sample was analyzed for Cr, Co, Bi, Hg, and Sb to see if any of them were present in sufficient quantity to serve as useful tracers. The results are shown on the attached report dated January 20, 1977, and marked invoice, number M 90076. All of the above, except Bi appear to be present in sufficient quantity that they can be analyzed for conveniently and could serve as useful tracers during ISL operations and may also prove helpful in exploration. These same elements are common pollutants and are under strict regulation by the EPA and state water quality boards. It is therefore recommended that the samples from this hole be analyzed for these elements and a profile developed. This information will be helpful as background data for environmental purposes to determine what horizons contain these elements and what their concentration is in case any claims of pollution are made in the future relative to any of these elements.

This same composite sample has been submitted for spectrographic analysis but the results are not yet available. On the basis of those results, future recommendations will be made regarding additional analyses which should be done.

Samples have not yet been selected for E-spec. It is recommended that samples from the intervals 430-433 feet, 418-419 feet, and 415-416 feet be submitted for E-spec to determine whether or not any other trace element indicators might be present which would be useful in exploration or in monitoring ISL operations. Other tracers might also be useful as environmental indicators if they can be tied to specific minerals or intervals, or if relationships can be established between them and other major pollutants.

It is recommended that more and fresher core be submitted for evaluation so that relationships alluded to in this report can be confirmed or discredited and generalizations drawn which can be helpful in exploration and ISL operations. Future reports will contain tables with correlation sufficient relating the various parameters.


Grant Buma, Consultant



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March 17, 1977

Nuclear Dynamics, Inc.
200 South Lowell
Casper, Wyoming 82601

ATTENTION: Mr. A.F. Stoick

SUBJECT: Sundance Project Laboratory
Batch Leaching Tests for Uranium Recovery

Dear Mr. Stoick:

Attached is our report of the laboratory batch leaching test for the Sundance project. This study involved seven batch leaching tests of drill core material from hole #829R using acid and carbonate lixiviants.

In compliance with your wishes at our Feb. 17th meeting, this report concludes the laboratory study. Samples from the batch leaching tests will be stored for at least six months. Thus, additional assays can be performed or reported values can be checked if desired.

I believe the information contained in this report will help clarify the process by which uranium is recovered for the Sundance project. If you have any questions in connection with the test results or if you would like additional study of any specific area, please be sure to call either Grant Buma or me.

Yours sincerely,

Frank J. Rosswog

Frank J. Rosswog
Engineering

FJR/nb

enc.

cc: T. G. Melrose

Summary

This report describes the laboratory study conducted to evaluate the leaching characteristics of the Sundance project uranium ore deposit. The study included seven batch leaching tests of a composite ore made up of core material from drill hole #829R. These tests determined the effects of carbonate, acid and peroxide additions to the leach liquor. Twenty-nine species or elements including U, V, Mo and Se, and pH were followed in each test.

The results of these tests indicate the following:

1. Carbonate leaching is superior to acid leaching in respect to uranium dissolution rate. The greater the carbonate-bicarbonate addition, the greater is the rate of dissolution.
2. Peroxide additions to the leach liquor even at low levels inhibits uranium dissolution.
3. Vanadium will dissolve and reprecipitate in both carbonate and acid leaches in the concentration ranges tested.
4. The effects of the dissolution of cations and anions other than uranyl complexes is minimized by the low level of mineralization of the ore deposit and the dilute nature of the leach liquor. In situ leaching will increase concentrations of vanadium, molybdenum and silica. Several other elements could also be slightly leached.

Data from the leach tests are organized into three tables. In addition, uranium and vanadium dissolution data are presented in six graphs.

SUNDANCE PROJECT
LABORATORY BATCH LEACHING TESTS
FOR URANIUM RECOVERY

Rocky Mountain Geochemical Corp.
1323 West 7900 South
Salt Lake City, Utah 84084
Phone (801) 255-3558

March 17, 1977

Investigators
F.J. Rosswog
B.G. Thomas

Introduction

The uranium leaching study and the tests described in this report were conducted by Rocky Mountain Geochemical Corp. at the Salt Lake City, Utah laboratory at the request of A.F. Stoick of Nuclear Dynamics, Inc. and T.G. Melrose of Bethlehem Steel W.U.S. This work is connected with the development of in-situ leaching of sandstone uranium deposits for the Nuclear Dynamics - Bethlehem Steel Sundance project.

The tests for the uranium leaching study consisted of seven batch leachings. Each batch was made up of one kilogram of dry ore and three liters of solution. Parameters varied between tests were in the chemical composition of the initial leach solution. Samples were drawn and analyzed as the leach progressed to determine rates of dissolution or precipitation. The over-all dissolution of several minerals was also determined by analysis of the heading and tailing ore of each batch test.

The test design for this study was aimed at providing relative data between many simplified leach tests. The simplicity of each test would allow accurate control of test conditions and completion of testing in a restricted time. The relative data from the tests would be used to demonstrate and evaluate the effects of varying several chemical parameters.

The batch leach design does not closely simulate the hydrological and geological characteristics of the in-situ leach and would not be a good basis for scale-up. The value of the batch leach design is that by its simplicity a wider range of parameters under rigidly controlled conditions can be evaluated. Then, by comparing data between batch tests, methods of improving the actual leach, potential leaching

problems and other leaching phenomena can be identified and evaluated. Batch leach testing is particularly effective at providing expedient data relative to primary product dissolution, effects of reagent additions, dissolution and precipitation of secondary products, and reagent consumption.

Batch Leaching Test Description

Each batch leaching test consisted of one kilogram of dry ore charge and three liters of initial leach liquor mixed in a four-liter nalgene plastic bottle. The ore charges for all tests were split from a common composite to insure uniformity between tests. This head composite was split from drill core material taken from hole #829R of the Sundance project. The head composite was dried at 40°C for 48 hours, reduced to -9 mesh (Tyler Scale) and blended for 30 min. before being split for each batch test.

Leach liquor for each batch test was drawn from a common 18-gal of formation water collected from the above drill hole. The hole was pumped 25 min. and then sampled over a 25 min. period. Additives for the leach solution - sodium carbonate, sodium bicarbonate, sulfuric acid and hydrogen peroxide - were of analytical grade.

Batch leach tests varied as to the amount of additive in the leach liquor. All tests were conducted at the same pressure and temperature - i.e., one atmosphere and 20°C.

Agitation of each batch test was provided by rotating the nalgene bottle on a small drum rotator. A common motor and shaft drove all bottles to give identical agitation to each. Rotation speed was 7.1 rpm. This mild agitation prevented establishment of concentration gradients, but was not severe enough to excessively grind the ore par-

ticles or emulsify the ore-liquid mixture.

The procedure followed in each batch test was as follows. The leach liquor was made up with the desired additives. The ore charge and leach liquor were rapidly and thoroughly mixed in each malgene bottle. The bottle was then continuously agitated on the drum rotator until 45 min. before sample time. At this time, agitation was stopped and the bottle was set upright to let the ore-liquor slurry settle. After 15 min. a 250-ml aliquant of solution was decanted and centrifuged for one hour. A sample of 150 ml of clear solution was then decanted from the 250-ml aliquant. The residue was returned to the leaching bottle. To make up for the leach liquor lost in sampling, 150 ml of fresh liquor with the initial concentration of additives was added at this time. The bottle was then vigorously agitated by hand and placed on the drum rotator until the next sample time.

At the end of a test, one fourth of the ore-liquor slurry was split, centrifuged and the clear liquor separated. After agitating with demineralized water, centrifuging and decanting the ore three times to separate residual leach liquor the remaining sample was dried and sent for analysis.

Batch Leaching Test Results

The analytical results from each batch leaching test are organized into three tables below (Tables 1, 2 and 3). Tests, test numbers and reagents added to the leach liquors correspond as follows:

<u>Test No.</u>	<u>Referred to as:</u>	<u>Reagent Additions</u>
1A	Formation Water	No. Na_2CO_3 or NaHCO_3 ;
	+ H_2O_2 Leach	.15 g/l H_2O_2
2A	1g/l Carbonate	1.0g/l $\text{CO}_3^{=}$ as a
	Leach	1:1 mole mix of

		Na ₂ CO ₃ and NaHCO ₃ ; .15 g/l H ₂ O ₂
3A	2g/l Carbonate Leach	2.0g/l CO ₃ ⁼ as in 2A; .15 g/l H ₂ O ₂
4A	2g/l Carbonate Leach, 2XH ₂ O ₂	2.0g/l CO ₃ ⁼ as in 2A; .15 g/l H ₂ O ₂
1B	4.0 pH Acid Leach	.3 ml/l Conc. H ₂ SO ₄ ; .15g/l H ₂ O ₂
2B	2.3 pH Acid Leach	.5ml/l Conc. H ₂ SO ₄ ; .15g/l H ₂ O ₂
3B	2.0 pH Acid Leach	.7ml/l Conc. H ₂ SO ₄ ; .15g/l H ₂ O ₂
4B	2.0 pH Acid Leach, 2XH ₂ O ₂	.7ml/l Conc. H ₂ SO ₄ ; .30g/l H ₂ O ₂
11	Formation Water Leach	No reagents added

Samples collected in each batch test are designated by the test number. Solution samples have an additional digit indicating the

sequence in which the samples were collected.

When examining the data in the tables the following should be noted. Arsenic and antimony were not determined in the solutions because of the low levels of the ore head sample. Silica interference with the sulfate determinations make the results somewhat questionable.

Uranium dissolution data are plotted in Figures 1, 2, 3 and 4. These data are grouped to show the relative effects of carbonate-bi-carbonate, acid and peroxide additions. For these figures, the uranium dissolution data are converted to uranium recovery by the following equation :

$$R_1 = \frac{3(C_1 - C_0) + .15 [(C_{1-1}C_0) + (C_{1-2}C_0) + \dots + (C_1 - C_0)]}{C_{ore}} \times 100 \quad (1)$$

where i = sample sequence number

R_1 = % uranium recovery at the time of
the i^{th} sample

C_1 = concentration of uranium in
the i^{th} sample

C_{ore} = concentration of uranium in the
unleached ore charge

Uranium recoveries can also be calculated from a material balance using the heading and tailing ore samples. These recoveries and the recoveries calculated for the last solution sample by the above equation are as follows:

<u>Test No</u>	<u>Uranium Recovery (%)</u>	
	<u>By Ore Material Balance</u>	<u>By Equation (1)</u>
1A	54	33
2A	60	42
3A	64	48
4A	61	43
1B	43	27
2B •	39	20
3B	29	20
4B	36	20
11 (Sample #11-4)	46	44

The uranium recoveries by the ore material balances are generally higher than by equation (1) because of the loss when washing the tailing ore sample with demineralized water to displace leach solution.

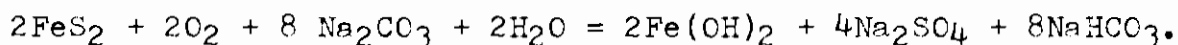
Reagent consumption for the leach tests having carbonate-bi-carbonate additions can be evaluated from a calculation of the combined carbonate as carbonation, $\text{CO}_3^{=}$, using values of Na_2CO_3 and Na_2HCO_3 from Table 1 and from a calculation of the carbonate-bicarbonate ratio. The calculated values for the combined carbonate-bicarbonate are:

<u>Test No</u>	<u>Combined Carbonate and Bicarbonate as $\text{CO}_3^{=}$ (g/l)</u>	
	<u>Initial Leach Solution</u>	<u>Final Leach Solution</u>
1A	.58	.69
2A	1.58	1.56
3A	2.58	2.46
4A	2.58	2.58
11	.58	.74

Comparison of initial and final values indicates that the net consumption of carbonate and bicarbonate is insignificant in these tests.

The ratio of carbonate to bicarbonate can be determined from the concentrations listed in Table 1. This ratio is also implied by

the pH of the solution. The ratio is about 1/10 at pH 9.3 for sodium carbonate and sodium bicarbonate in demineralized water. An increase of one pH unit increases the ratio by a factor of ten; a reduction of one unit decreases it by ten. Thus, the decline of pH during each carbonate leach indicates a consumption of carbonate and a production of bicarbonate. The major cause of this shift in ratio is the oxidation and dissolution of sulfides from the ore. The overall reaction with carbonate consumption and bicarbonate production is:



The dissolution of alumina and silica can also contribute to the shift in carbonate-bicarbonate ratio.

The oxidation of sulfides and the resulting reduction in pH is not necessarily detrimental to uranium leaching. A lower ratio of carbonate to bicarbonate can improve uranium dissolution rates and keep calcium carbonate from precipitating. However, high concentrations of sulfate in solution will precipitate calcium sulfate (gypsum) with the result of fouling at leaching channels. Sulfates also compete in the ion exchange absorption of uranium and will reduce the efficiency of the recovery system. Incomplete oxidation of sulfides produces polythionates which are strong chemical poisons for the resin. The dilute nature of the leach solution and low mineralization of the ore deposit will probably prevent sulfates and polythionates from becoming a significant problem.

An evaluation of reagent consumption for acid leaching using data from those leach tests having sulfuric acid additions is complicated by the need to neutralize the original formation water.

Once the initial neutralization is completed, acid consumption would drop considerably. Care must also be taken to prevent precipitation of calcium sulfate and the plugging of leaching channels.

The dissolution of cations, in general, does not cause difficulty in the recovery of uranium with ion exchange resin. The light metals, sodium and potassium, do not form carbonate and sulfate complexes as does uranium and, thus, are not loaded on the resin. In carbonate solutions, these cations are not dissolved to any great extent. Acid solutions, however, will mobilize these cations. The precipitation of calcite can cause surface fouling of the resin. However, at the low concentrations of calcium measured in the batch leach tests this would not seem to be a problem.

Anions and anion complexes other than the uranyl carbonate complex in carbonate solutions and the uranyl sulfate complex in acid solutions are deleterious to some extent in all cases to the recovery of uranium in that anions compete for resin sites. These anions come from the dissolution of the heavy metals and non-metals. The major constituents encountered are chloride, fluoride, sulfate, aluminate, phosphate, molybdate and vanadate. Chloride, fluoride and sulfate can be removed with normal elution and do not poison the resin. Their effect is proportional to their concentration in solution. Aluminate, phosphate and molybdate can polymerize and precipitate with changes in pH causing mechanical poisoning of the ion exchange resin. The low concentration of phosphate in the ore deposit should preclude this anion being a problem. Molybdate is less strongly absorbed from carbonate solutions and because of its low concentration will probably not be a problem. Vanadate anion complexes are strongly absorbed by

the ion exchange resin in preference to uranium and, thus, are a problem in uranium recovery. The reprecipitation of vanadium (see Figures 5 and 6) may prevent any build-up in solution. Also, vanadium can be selectively eluted from the resin or in other ways separated from the uranium product.

Silica, polythionates and sulfur can also cause problems in ion exchange circuits. Silica dissolves as a non-ionic species at low pH (compare acid and carbonate leach tests on Tables 1 and 2) and will precipitate as a polymeric gel at higher pH. This gel can cause severe fouling in both leaching and ion exchange recovery. Polythionates and sulfur can be produced from the incomplete oxidation of sulfide minerals. Polythionates are strongly absorbed by the resin. Sulfur can cause mechanical fouling. Because of the low sulfur content of the ore (Table 3), polythionates and sulfur will probably not be a problem in the ion exchange circuit.

The results of the batch leaching tests indicate the impact in-situ leaching of uranium will have on the ground waters of the formation. Some of the heavy metals--chromium, cobalt, copper, mercury and bismuth--are not present in the deposit or do not dissolve to any appreciable extent in either carbonate or acid leach solutions (see Tables 1, 2 and 3). Uranium and vanadium dissolve rapidly; uranium continues to build up while vanadium reprecipitates (see Figures 1, 2, 3, 4, 5 and 6). Vanadium is still many times its original level at the completion of a test. Molybdenum in both acid and carbonate leaches shows a concentration increase of a factor of ten to a steady level after four hours of leaching. Manganese increases in concentration in acid solutions but not in carbonate solutions. Iron and aluminum show some slight increases.

Of the light metals, sodium concentrations reflect the addition of sodium carbonate and sodium bicarbonate. Potassium increases by four to five times in both acid and carbonate solutions. Acid solutions produce increases in magnesium and calcium while carbonate leaching causes some decrease in calcium.

In general, non-metals not including the halogens increase in concentration. Selenium is an exception being of a very low level in the ore. Phosphorus and sulfur as sulfate increase slightly in each test. Silica doubled in the carbonate leaching and increased to a slightly greater extent in the acid leaching. The halogens, fluorine and chlorine, remain constant in the carbonate leaching and both decrease slightly in the acid leaching.

An indicator of the overall dissolution of the ore deposit is the measurement of the total dissolved solids (TDS) of the leach liquor. This measurement is given in the last column of the tables. The values roughly reflect the additions made to the original leach solution and indicate no gross build-up of dissolved solids.

Conclusions and Recommendations

Several general conclusions can be made from examining the above batch leaching results. Uranium dissolution in leach liquors having carbonate-bicarbonate additions exceeds that in liquors having acid additions in the ranges tested (see Figures 1 and 3). The more carbonate-bicarbonate added the faster was the rate of dissolution; the more acid added, the more inhibited was the rate of dissolution. The relative ineffectiveness of acid leaching can be attributed to the amount of magnesium and calcium in the core material-- .33 and .50%,

respectively. As would be expected and as seen in the pH rise of the acid leach tests, the core material rapidly neutralizes any acid in the leach liquor.

Peroxide additions in both acid and carbonate leaching inhibits uranium dissolution. This effect should not be attributed to any lack of potential of the peroxide to oxidize the ore mineralization but to the fact that the peroxide is an effective precipitant of hydrated uranium peroxide. Evidently, even at the low concentrations of peroxide used, the precipitation effect is greater than any improvement because of oxidation. Hydrogen peroxide was used because of the ease of handling. A less vigorous oxidizer that would not form uranium peroxide could give improved rates of uranium dissolution.

Vanadium reprecipitates in both acid and carbonate leaches in the range of parameters tested. This reprecipitation is evidenced by the rapid buildup of vanadium during the early part of each test followed by a more gradual decline. Reprecipitation of vanadium could be disadvantageous to uranium dissolution because coprecipitation of uranium and vanadium could be taking place as is common in some leaching circuits.

In general, the test results are good in respect to good resolution between tests and providing data on dissolutions and precipitations. This data can be used as part of a preliminary study in preparation of large-scale on-site leaching tests.

A continuation of laboratory batch testing could include some of the following refinements. Other oxidizers could be evaluated including metered amounts of air and oxygen. Ore charges could be prepared without drying the ore and with minimum exposure of the ore to air. In this way, oxidation and other alterations of the ore could be

reduced. Tests could also be conducted under pressure to more closely simulate in-situ leaching conditions. Any of these refinements would add complexity to the testing procedure or increase the difficulty in preparing the ore charge. Therefore, the value of the resulting data should be weighed carefully keeping in mind the limitations of reproducing in-situ leaching conditions.

Further laboratory studies should be conducted to determine effective means of dealing with residual U, V, Mo, etc. after the leaching operations have been completed.

TABLE I
Analysis of Carbonate Leach Liquors

Sample	Test and Sample Description	2500 (ppm)	V ₂ O ₅ (ppm)	Mo (ppm)	Se (ppm)	Cr (ppm)	Mn (ppm)	Fe (ppm)	Co (ppm)	Cu (ppm)	Hg (ppm)	Al (ppm)	Si (ppm)	pH
Head	Heading Solution from Formation	1	<.4	<.02	<.04	.32	<.1	.4	<.1	<.04	<.1	.3	4.2	8.6
1A-1	Formation Water + H ₂ O ₂ Leach at 2 hrs	14	5.4	.24	—	—	—	.4	—	—	—	1.1	7.7	8.5 (Initial pH 8.0)
1A-2	" " 6 hrs	19	4.3	.24	—	—	—	.8	—	—	—	.8	7.0	8.3
1A-3	" " 14 hrs	21	1.8	.28	—	—	—	.7	—	—	—	1.1	9.0	8.4
1A-4	" " 32 hrs	24	.7	.24	—	—	—	.6	—	—	—	.5	7.0	8.2
1A-5	" " 57 hrs	28	.7	.24	<.04	.40	<.1	1.4	<.1	<.04	<.1	.5	6.3	8.3
2A-1	1g/l Carbonate Leach at 2 hrs	22	6.4	.32	—	—	—	1.2	—	—	—	.8	7.7	9.2 (Initial pH 9.2)
2A-2	" " 6 hrs	26	6.1	.28	—	—	—	1.4	—	—	—	1.1	8.4	9.3
2A-3	" " 14 hrs	27	4.3	.26	—	—	—	1.0	—	—	—	1.9	9.8	9.1
2A-4	" " 32 hrs	34	3.9	.28	—	—	—	2.2	—	—	—	1.1	9.0	9.2
2A-5	" " 57 hrs	35	3.6	.26	<.04	.36	<.1	1.0	<.1	<.04	<.1	—	—	9.2
3A-1	2g/l Carbonate Leach at 2 hrs	24	6.4	.30	—	—	—	1.2	—	—	—	.8	7.0	9.5 (Initial pH 9.8)
3A-2	" " 6 hrs	26	6.1	.30	—	—	—	.6	—	—	—	.3	7.7	9.5
3A-3	" " 14 hrs	31	5.0	.26	—	—	—	.8	—	—	—	.3	7.7	9.5
3A-4	" " 32 hrs	34	5.0	.24	—	—	—	.6	—	—	—	.3	8.4	9.5
3A-5	" " 57 hrs	40	4.6	.28	<.04	.36	<.1	.6	<.1	<.04	<.1	.3	7.7	9.5
4A-1	2g/l Carb. Leach, 2x H ₂ O ₂ at 2 hrs	21	6.8	.28	—	—	—	.6	—	—	—	.5	7.7	9.5 (Initial pH 9.8)
4A-2	" " 6 hrs	26	6.8	.28	—	—	—	1.2	—	—	—	.3	7.0	9.5
4A-3	" " 14 hrs	Sample lost during centrifugation										—	—	—
4A-4	" " 32 hrs	32	5.0	.26	—	—	—	1.4	—	—	—	.3	8.4	9.5
4A-5	" " 57 hrs	36	4.6	.24	<.04	.32	<.1	1.6	<.1	<.04	<.1	.3	7.7	9.5

TABLE 1 (Cont'd)
Analysis of Carbonate Leach Liquors

Sample	Na_2CO_3 (g/L)	NaHCO_3 (g/L)	FeO (ppm)	Bi (ppm)	SO_4^{2-} (g/L)	Na (ppm)	K (ppm)	Mg (ppm)	Ca (ppm)	Cl (ppm)	F (ppm)	TDS (g/L)
Head	.032	.77	<1	<1	.88	700	4.0	3.6	20.5	21.3	.48	1.82
1A-1	.11	.71	<1	—	1.00	600	11.5	5.0	26.5	17.7	.72	—
1A-2	.11	.84	—	—	1.04	600	12.5	5.5	31.0	19.9	.75	—
1A-3	nil	.101	—	—	1.04	600	12.0	6.0	31.5	19.0	.66	—
1A-4	.11	.88	—	—	1.01	600	11.0	5.5	29.0	19.9	.70	—
1A-5	.11	.88	<1	<1	1.05	600	12.0	6.0	33.0	19.9	.75	2.09
2A-1	.42	.60	<1	—	1.03	1100	12.5	6.5	12.5	28.4	.76	—
2A-2	.32	.85	—	—	1.05	900	12.0	6.0	12.0	27.1	.86	—
2A-3	.32	.85	—	—	1.08	900	13.0	5.5	14.0	34.6	.84	—
2A-4	.32	.85	—	—	1.05	900	11.0	6.0	11.0	26.2	.79	—
2A-5	.32	.93	1	<1	1.05	900	10.5	6.0	11.5	26.2	.81	2.98
3A-1	.106	2.60	<1	—	1.01	1150	13.5	10.5	10.0	26.2	.76	—
3A-2	.106	2.60	—	—	1.07	1150	13.0	10.0	8.0	27.1	.83	—
3A-3	.106	2.60	—	—	1.06	1150	13.0	9.5	8.5	28.0	.76	—
3A-4	.117	2.44	—	—	1.06	1100	13.0	9.5	8.5	27.1	.81	—
3A-5	.106	2.60	3	<1	1.11	1250	14.5	10.0	9.5	24.4	.78	4.23
4A-1	.106	2.60	2	—	1.07	1100	13.5	10.5	11.5	32.0	.73	—
4A-2	.106	2.60	—	—	1.04	1100	13.5	11.0	10.5	28.4	.83	—
4A-3	—	—	—	—	—	—	—	—	—	—	—	—
4A-4	.106	2.60	—	—	.98	1100	14.0	10.0	9.0	28.0	.75	—
4A-5	.106	2.77	2	<1	1.02	1250	14.0	9.5	10.0	28.4	.72	4.21

TABLE 2
Analysis of Acid Lach Liquors.

[illegible]

TABLE 2 (Cont'd)
Analysis of Acid Leach Liquors

[illegible]

Figure 1

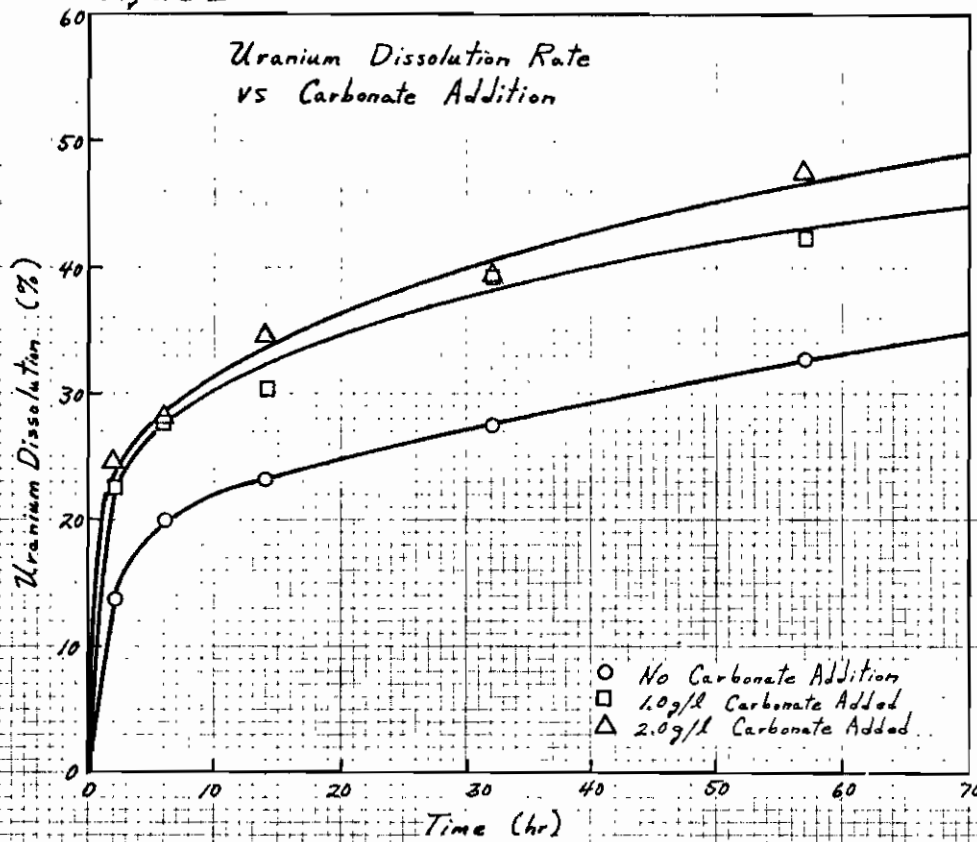


Figure 2

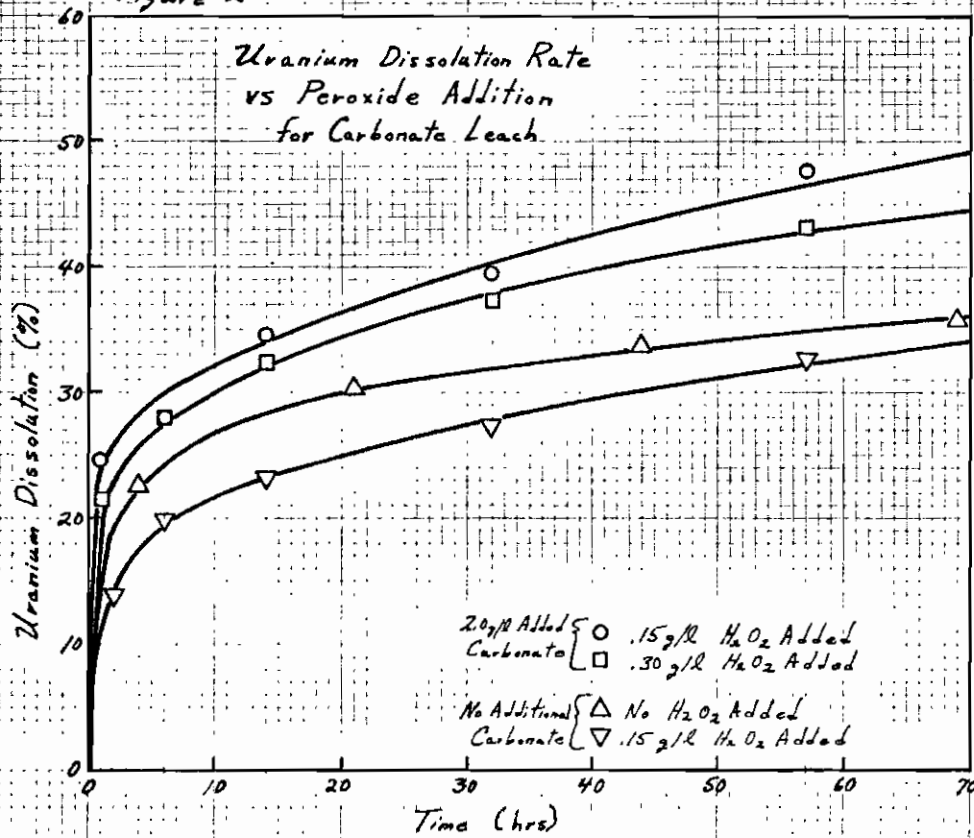


Figure 3

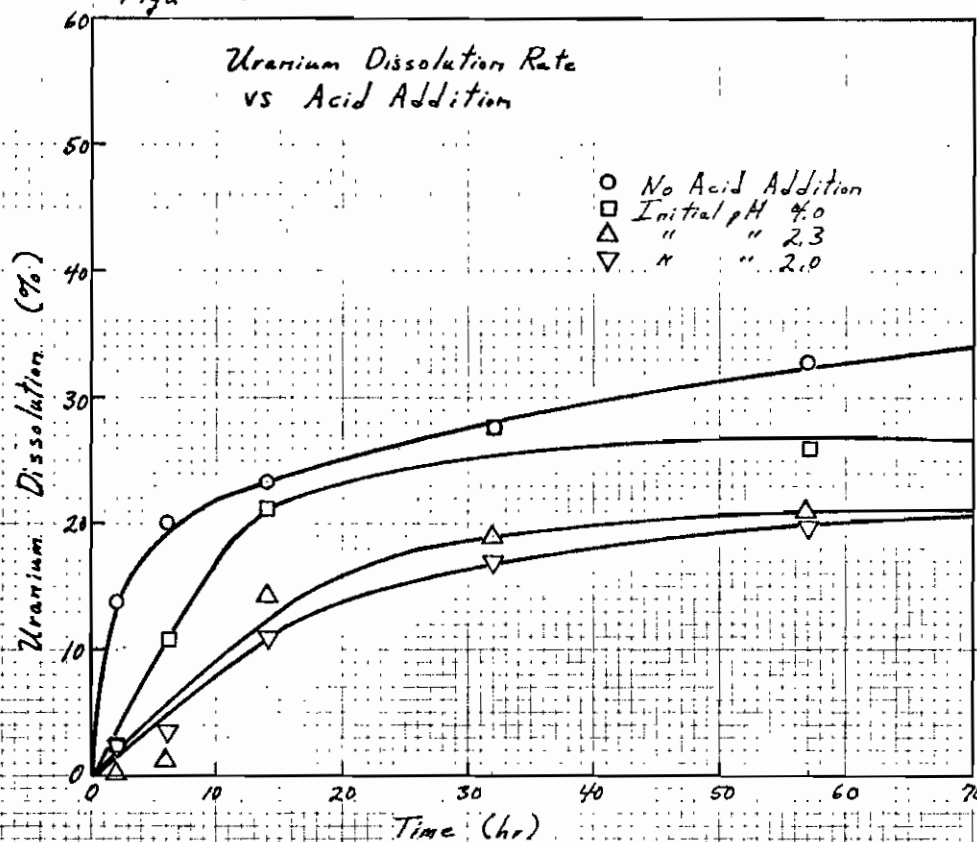


Figure 4

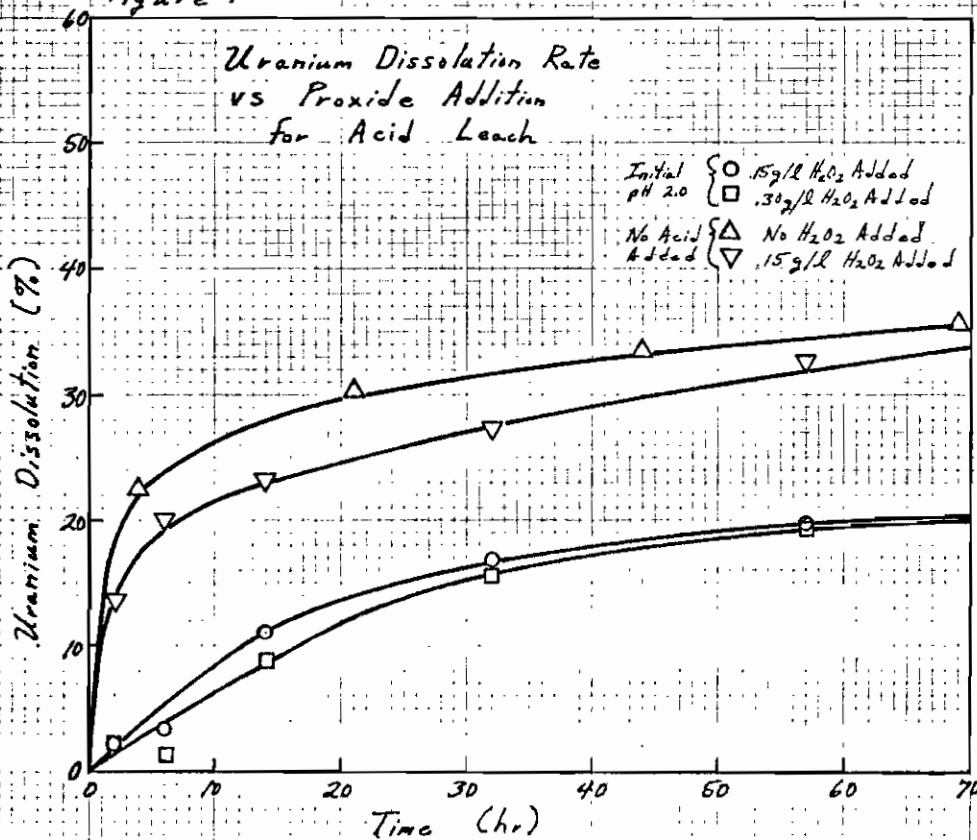


Figure 5

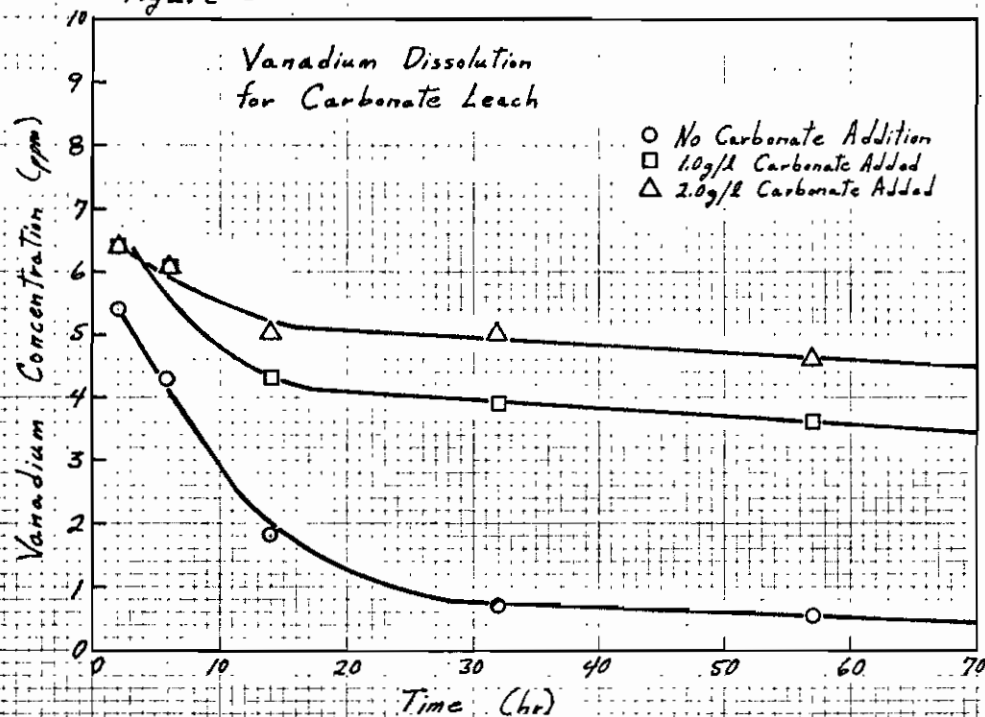
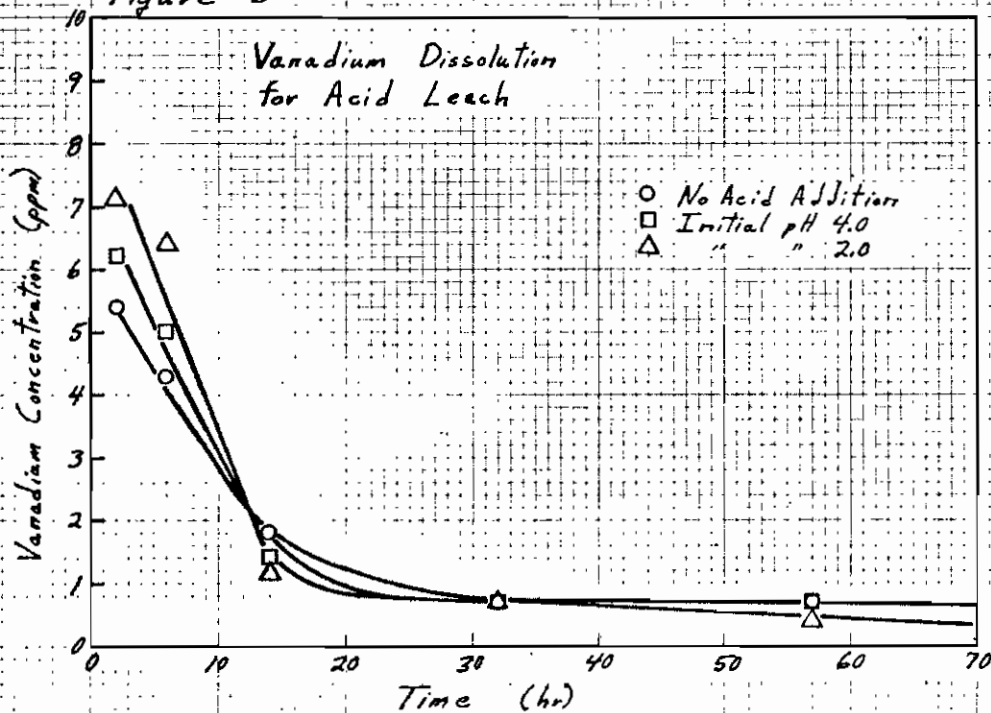


Figure 6





MIDVALE OFFICE

ROCKY MOUNTAIN GEOCHEMICAL CORP.

P. O. BOX 337 • 1323 W. 7900 SOUTH • MIDVALE, UTAH 84047 • PHONE: (801) 255-3558

Certificate of Analysis

Page 1 of2.....

Date: March 11, 1977
Client: Nu-Beth
P. O. Box 337
Midvale, Utah 84047

RMGC Numbers:

Local Job No.: 77-33-28-SL

Foreign Job No.:

Invoice No.: M 90274

Client Order No.: none
Report On: 11 samples
Submitted by: Grant Buma
Date Received: 12/29/76
Analysis: Total Dissolved solids
Analytical Methods: determined gravimetrically.

Remarks:

cc: enc.
file (2)
LRR/lw

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND == None Detected 1 ppm == 0.0001% 1 Troy oz./ton == 34.286 ppm 1 ppm == 0.0292 Troy oz./ton

<u>Sample No.</u>	<u>g/l</u> <u>Total Dissolved Solids</u>
829-R	1.79
1A 5	2.09
2A 5	2.98
3A 5	4.23
4A 5	4.21
1B 5	2.60
2B 5	2.98
3B 5	3.54
4B 5	3.61
#11 69hr	2.06
Heads	1.82

By Lawrence R. Reid
Lawrence R. Reid

**ROCKY MOUNTAIN GEOCHEMICAL CORP.**

SALT LAKE CITY UTAH

RENO NEVADA

TUCSON ARIZONA



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Certificate of Analysis

Page 1 of 3

Date: Feb. 28, 1977
Client: Nu-Beth
P. O. Box 337
Midvale, Utah 84047

RMGC Numbers:
Local Job No.: 76-33-28-SL
Foreign Job No.:
Invoice No.: M 90204

Client Order No.: none

Report On: 32 Samples

Submitted by: Grant Buma

Date Received: 1/18/77

Analysis: U_3O_8 , V_2O_5 , Molybdenum, and Fluorine

Analytical Methods: V_2O_5 and Molybdenum determined by atomic absorption.
 U_3O_8 determined by Fluorometric. Fluorine determined by specific ion electrode.

Remarks:

cc: enc.
file (2)
LRR/lw

Sample No.	ppm U_3O_8	ppm V_2O_5	ppm Molybdenum
tailings 11	150	130	-1

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND = None Detected 1 ppm = 0.0001% 1 Troy oz./ton = 34.286 ppm 1 ppm = 0.0292 Troy oz./ton

<u>Sample No.</u>	<u>ppm Fluorine</u>
1A-1	.72
2	.75
3	.66
4	.70
5	.75
2A-1	.76
2	.86
3	.84
4	.79
5	.81
3A-1	.76
2	.83
3	.76
4	.81
5	.78
4A-1	.73
2	.83
3	
4	.75
5	.72
1B-1	.18
2	.29
3	.24
4	.32
5	.28



<u>Sample No.</u>	<u>ppm</u> <u>Fluorine</u>
2B-1	.30
2	
3	
4	
5	.38
3B-1	.18
2	
3	
4	
5	.24
4B-1	.13
2	
3	
4	
5	.24
heads	.48
formation	.54

By Lawrence R. Reid
Lawrence R. Reid





ROCKY MOUNTAIN GEOCHEMICAL CORP.

MAILING ADDRESS:
P. O. BOX 337
MIDVALE, UTAH 84047

BUSINESS LOCATION:
1323 WEST 7900 SOUTH
WEST JORDAN, UTAH 84084
PHONE: (801) 255-3558

February 25, 1977

Nuclear Dynamics Corp.
Casper, Wyoming

Attention: Mr. Albert Stoick, Manager

Subject: Direct Uranium Precipitation Study for Sundance In
Situ Leaching Tests

Dear Mr. Stoick:

Enclosed is a copy of our report describing the preliminary laboratory tests that have been conducted in connection with direct uranium precipitation. These laboratory tests were initiated at a request by Tom Melrose. Presently, Tom Melrose has asked us to discontinue any laboratory investigations and conclude the study with the preparation of the enclosed report. Because of the desire to stop work at this point, the report is far from being thorough or complete. However, it does summarize the testing completed to date.

If you have any questions, please be sure to call me or Grant Buma.

Yours sincerely

Frank J. Rosswog
Engineering

FJR/nb

cc: Tom Melrose

Summary

This report describes a series of tests of direct precipitation techniques for use with in-situ leaching of uranium. These tests were designed to quickly evaluate feasibility and cover the following methods:

1. Precipitation with lime
2. Degassing Followed by Precipitation with Lime
3. Precipitation of Uranous Phosphate
4. Precipitation with Ammonium meta-Vanadate
5. Precipitation by Reduction with Sodium Sulfide

The laboratory tests produced the following results. Method 2 using combined acidification and vacuum degassing and methods 3 and 4 are promising techniques with precipitation efficiencies at greater than 92%. Also for method 3, excess soluble phosphate can be stripped from the leach liquor with lime. The other methods tested did not have high precipitation efficiencies.

Because of the restrictive and incomplete nature of the above study, further laboratory scale testing is recommended before a specific technique is chosen for systematic and extensive development. However, both methods 2 and 3 could be developed rapidly for expedient in-situ leaching tests.

PRELIMINARY LABORATORY EVALUATION
OF DIRECT PRECIPITATION METHODS
FOR IN-SITU LEACHING OF URANIUM

Rocky Mountain Geochemical Corp.
1323 West 7900 South
Salt Lake City, Utah 84084
Phone (801) 255-3558

Date: March 1, 1977

Investigator:
Frank J. Rosswog
Engineering

Introduction

The test program of which this test report is a part was initiated by a request by T.G. Melrose, Bethlehem Steel Corp., for Rocky Mountain Geochemical Corp. to investigate the possibility of directly precipitating uranium from a leach liquor produced by the Sundance project in-situ leach tests. The resulting product of the direct precipitation would be amenable to sale, further processing and uranium recovery.

The following preliminary laboratory tests evaluate eight promising methods of direct precipitation. Each test is designed to give a quick answer as to the amenability of a particular method to treating in-situ leach liquor. Subsequent testing of those methods with positive results would determine reagent consumption and reaction kinetics.

The direct precipitation methods tested and reported herein are:

1. Precipitation with Lime
2. Degassing Followed by Precipitation with Lime
3. Precipitation of Uranous Phosphate
4. Precipitation with Ammonium meta-Vanadate
5. Precipitation by Reduction with Sodium Sulfide

Results of this testing are summarized in Table 1.

Precipitation with Lime

A simple flow scheme for precipitating uranium is to mix in milk-of-lime with the process stream. The hydrated calcium oxide of the milk-of-lime, a strong base, when added to a carbonate-bi-carbonate solution will first react with the bicarbonate to form

TABLE 1
Uranium Precipitation Tests

Test Description	Solution Uranium Concentration Initial (ppm) Final (ppm)		Final pH	Remarks
Precipitation with Lime	39	49	9.8	Evaporation losses of about 20% caused uranium levels to increase Precip. Eff. 98%
" " " " , 20g Lime Added	39	45	9.8	
Acid Leach Liquor	22	.5	9.5	
Degassing Followed by Precipitation with Lime	33	33	9.0	Precip. Eff. 29%
	33	33	9.5	
	33	34	9.5	
	33	27	9.2	
	41	29	9.9	
Vacuum, Lime Added	41	38	12.0	Precip. Eff. 97%
" , NaOH Added	38	1.1	9.4	
Precipitation of Uranoous Phosphate	32	2.7		Precip. Eff. 92%
Monobasic Potassium Phosphate	65	50		
Orthophosphoric Acid, 31ppm PO ₄	65	61		
" " , 34ppm PO ₄	65	65		
" " , 62ppm PO ₄	65	3.0		
" " , 108 ppm PO ₄				Precip. Eff. 95%, Final [PO ₄] 1.5ppm
Precipitation with Ammonium meta-Vanadate	32	1.7		Precip. Eff. 95%
Precipitation by Reduction with Sodium Sulfite	32	24		Precip. Eff. 25%

insoluble calcium carbonate. As more base is added and the pH is increased, uranium is hydrolyzed and precipitates as a mixture of diuranates and polyuranates.

To test this method of precipitation, three trials each with 500 ml aliquants of leach liquor were evaluated. The first aliquant was a 2 g/l carbonate liquor from laboratory leach tests of drill core material (Hole #829R). Initial uranium concentration of the leach liquor was 39 ppm before precipitation. To this first aliquant was added 1.0 g of commercial lime. After stirring for 25 hrs., the liquor was sampled and the uranium content determined to be 43 ppm. The increase in concentration was because of evaporation loss. The pH was 9.8 and the concentration of U_3O_8 in the solids was .05%. Additional lime was added to the liquor, after 13 hrs. the liquor was again sampled and the uranium concentration determined. The uranium concentration, 49 ppm, indicated no decrease; thus, this trial had ineffective direct precipitation.

The second 500 ml aliquant was of leach liquor identical to the first and was treated similarly but with 2.0 g of commercial lime added. The results were 47 ppm U_3O_8 in solution after 25 hrs., pH 9.8, and solids containing .03% U_3O_8 . Additional lime reduced the uranium concentration to only 45 ppm U_3O_8 .

The third 500 ml aliquant was of acidified leach liquor from a similar leaching test as the carbonate leach test but with initial formation water acidified with sulfuric acid to pH 2.0. This liquor contained 22 ppm U_3O_8 before precipitation. An addition of 1.1 g of commercial lime, enough to increase the pH to 9.5, reduced the uranium content of the liquor to 4 ppm U_3O_8 in

4 hrs. The precipitate contained .86% U_3O_8 . A further addition of lime reduced the uranium level to .5 ppm U_3O_8 .

The above test results are summarized in Table 1. These tests indicate that direct precipitation with lime does not work with carbonate leach liquors. Acid leach liquors or, perhaps, leach liquors in which the carbonates and bi-carbonates have been destroyed can be treated by the above technique.

Degassing Followed by Precipitation with Lime

An improved flow scheme for carbonate leach liquors as indicated by the above trials would be the addition of a carbonate destruction and removal step before precipitation with milk-of-lime. This initial step could be accomplished in a variety of ways among which are the following:

Acidification

Adding sulfuric acid to the leach liquor would destroy the bicarbonates. At pH less than 3, CO_2 starts to effervesce. Because of the considerable solubility of CO_2 in water, complete removal is difficult.

Vacuum degassing

Because of the fact that a carbonate-bicarbonate solution is in equilibrium with a finite CO_2 pressure above the solution, CO_2 can be removed and carbonate-bicarbonate destruction accomplished by subjecting the solution to vacuum. At low pressures, however, the presence of bases in the solution inhibit this effect.

Combined Acidification and Vacuum Degassing

Combining the above two methods would overcome the shortcomings of each.

Air Sparging

Air sparging has much the same effect as vacuum degassing. Because of the low CO_2 vapor

pressure of the air, CO_2 is stripped from the solution. In general, this method is slow at decreasing carbonate and bicarbonate strengths and, as the pH increases, the stripping is inhibited.

Thermal Decomposition

For many common solutions, heating reduces the solubility of gases. This is true for a carbonate-bicarbonate solution which can be viewed as dissolved CO_2 . Thus, by heating, a CO_2 -rich vapor can be formed and separated from the solution, reducing the carbonate-bicarbonate concentration. Again, as in the cases of air sparging and vacuum degassing, bases inhibit this effect.

The above are some of the most common and easily applied methods of destroying carbonates and bicarbonates. Preliminary laboratory tests of the first three of these methods in conjunction with direct uranium precipitation with lime were conducted. These tests are described below.

To test the technique of acidification with sulfuric acid, four trials each with a 500 ml aliquant of carbonate leach liquor from laboratory test leaching of drill core material (Hole #829R) were conducted. Each aliquant was acidified and stirred vigorously for one hour. Amounts of lime estimated to increase the pH to between 9.0 and 9.5 were added to each aliquant. After 14 hrs., the solutions were sampled and the uranium concentrations determined. Test parameters showing pH on acidification and amounts of lime added and the test results are summarized in Table 1. These trials and the test results indicate that acidification alone does not improve the direct precipitation with lime to a great extent.

Degassing with vacuum alone was tried with two 500 ml aliquants of carbonate leach liquor. Each aliquant was subjected to a vacuum of 22 in. Hg for 30 min. Considerable effervescence was observed during the application of the vacuum. Following degassing, one aliquant was treated with 10 g of lime; the other was treated with 1 g of sodium hydroxide.

The results are summarized in Table 1 and indicate about 30% of the uranium can be precipitated by this technique using lime. Sodium hydroxide was not effective as a precipitant. A 30% precipitation of uranium would not be adequate for direct precipitation in an in-situ leaching operation.

A combination of acidification and vacuum degassing was evaluated with one trial using a 500 ml aliquant of carbonate leach liquor. Sulfuric acid was added to reduce the pH to about 2 with no effervescence being noted. On reducing the pressure above the solution to 22 in. Hg vacuum the degassing was rapid and was completed in 5 min. Precipitation was accomplished by increasing the pH of the solution to 9.4 with lime and stirring for 4 hours.

Results are summarized in Table 1 and show that this combined technique gives a high precipitation efficiency of 95%.

Precipitation of Uranous Phosphate

One of the early and one of the most selective techniques for precipitating of uranium is the precipitation of uranous phosphate. This method is also suggested as a technique for direct precipitation by the presence of autunite, $\text{Ca (U O}_2)_2(\text{PO}_4)_2 \cdot 10-12 \text{ H}_2\text{O}$, deposits.

The source of phosphate used in early precipitation methods

was any soluble ortho- or phrophosphate. For the trials reported below, monobasic potassium phosphate and ortho phosphoric acid were evaluated; orthophosphoric acid being the cheapest source.

To regenerate the leach liquor for in-situ leaching, any excess soluble phosphate must be removed or precipitation of uranous phosphate will occur during leaching. To accomplish this removal of excess phosphate, the uranium precipitation is followed by stripping the leach liquor with lime. Most calcium phosphates have relatively low solubility.

The first trial involved treating a 300 ml aliquant of carbonate leach liquor containing 32 ppm of U_3O_8 with 1.0 g of KH_2PO_4 , an amount in excess of that required to stoichiometrically precipitate the uranium and calcium. The calcium in solution being 20 ppm, more than enough to form autunite. The uranium precipitation in this trial was a promising 92%. A summary of the test parameters and results for this and the following tests are presented in Table 1.

A set of four trials each of a 500 ml aliquant of carbonate leach liquor and using orthophosphoric acid as a precipitant was conducted. Initial concentrations of phosphate expressed as P_2O_5 for each of these trials were 31, 34, 62 and 308 ppm. These concentrations are the amounts estimated to be 100%, 110%, 200% and 10 times, respectively, that required for stoichiometric precipitation. The last and highest concentration reduced the uranium by 95% in less than 4 hrs. To evaluate the ability of lime to strip excess soluble phosphates from the liquor, the aliquant from the last trial was split three ways and lime to the amounts of

.5 g/l, 2 g/l and saturation (about 4 g/l) was stirred in. Samples of solution were separated after two hours and the phosphate concentrations were determined to be 176, 116 and 1.5 ppm as P_2O_5 . Initial phosphate concentration before lime stripping was 220 ppm.

The above results indicate that a uranous phosphate precipitation followed by lime stripping is a practical method for direct precipitation of in-situ leach liquors. The 95% precipitation efficiency is adequate for in-situ leaching and a phosphate concentration of 1.5 ppm P_2O_5 in the return leach liquor is less than the 4-5 ppm found in liquors from laboratory leach tests.

Precipitation with Ammonium meta-Vanadate

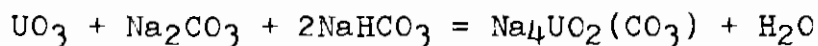
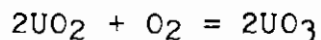
The coprecipitation of Vanadium and Uranium forms the well known mineral carnotite, $K_2(UO_2)_2(VO_4)_2 \cdot 1-3 H_2O$. This same coprecipitation can be used to directly precipitate uranium from carbonate-bicarbonate leach liquors by adding a soluble vanadium salt such as ammonium meta-vanadate.

To test this technique for the in-situ leach liquor, a 300 ml aliquant of carbonate liquor from laboratory leach tests was treated with 1.0 g ammonium meta-vanadate. After 24 hrs. the solution was sampled and the uranium concentration determined.

The results of this trial are summarized in Table 1 and show that the uranium precipitation is effective. However, the high cost of soluble vanadium salts and the difficulty of removing excess vanadium salts from the leach liquor before return to in-situ leaching probably precludes this technique as a practical method of direct uranium precipitation.

Precipitation by Reduction with Sodium Sulfide

The overall reactions for the dissolution of uranium oxide in a carbonate-bicarbonate lixiviant are:



A depletion of oxygen in the above system by adding a reducing agent such as sodium sulfide would shift equilibrium to the left and cause UO_2 to precipitate. Such a reduction technique might be practical for direct precipitation from in-situ leach liquors. However, the kinetics of such a reversed reaction are not so easily understood and are not implied by the simple overall equations.

The test described below gives a rapid evaluation of this reduction technique. A 300 ml aliquant of carbonate-bicarbonate leach liquor was treated with 1.0 g of sodium sulfide ($\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$). After being mixed for 24 hours, the solution was sampled and the uranium concentration determined.

The results indicate a 25% reduction in soluble uranium. In practice, this reduction would not be adequate for in-situ leaching. However, stronger and, consequentially, more difficult to handle, reducing agents could give more promising results.

Conclusion

The above direct precipitation trials evaluate some of the more practical techniques. These trials do not, however, exhaust the list of methods that could be evaluated. Additional testing might include thermal and electrolytic decomposition of carbonates, thermal hydrolysis, arsenate precipitation, hydrogen reduction with a catalyst, and hydrated uranium peroxide precipitation. Many

of these additional methods represent considerable additional complexity to the in-situ leaching operation and could be discarded for this reason. The preliminary evaluations reported above include several of the simpler methods for direct precipitation.

For a continuation of an investigation into direct precipitation techniques, further laboratory scale testing is recommended before a specific method is selected for extensive development. For an expedient in-situ leaching study, of the above methods tested either the combined technique of acidification and vacuum degassing followed by precipitation with lime or the phosphate precipitation followed by lime stripping could be developed rapidly.



ROCKY MOUNTAIN GEOCHEMICAL CORP.

MAILING ADDRESS:
P. O. BOX 337
MIDVALE, UTAH 84047

BUSINESS LOCATION:
1323 WEST 7900 SOUTH
WEST JORDAN, UTAH 84084
PHONE: (801) 255-3558

February 25, 1977

Uranium Exploration and Management
Nuclear Dynamics, Inc.
200 So. Lowell
Casper, Wyoming 82601

Attention: Mr. A. F. Stoick

Subject: Sundance Project Uranium Recovery Pilot Plant Costs

Dear Mr. Stoick:

This letter is in response to your request for our bid on the 50 gpm ion exchange pilot plant that would be suitable for the Sundance in situ leach test program. Our bottom line selling price for an assembled plant with equipment checked for operation and ready for partial disassembly and shipment is \$120,000. A cost plus bid for the same plant is less by a 15% contingency--i.e., \$104,000. With these bids we could add a \$50/day mulct for any construction requiring more than 101 days to complete and a similar premium for finishing construction before 91 days.

Attached is a breakdown of the costs for constructing the plant and a diagram showing the allotted time.

I hope this information will meet your requirements. If you have any questions, please call either Frank Rosswog or me.

Yours sincerely,

ROCKY MOUNTAIN GEOCHEMICAL CORP.

Grant Buma, President

GB/lw

enc.

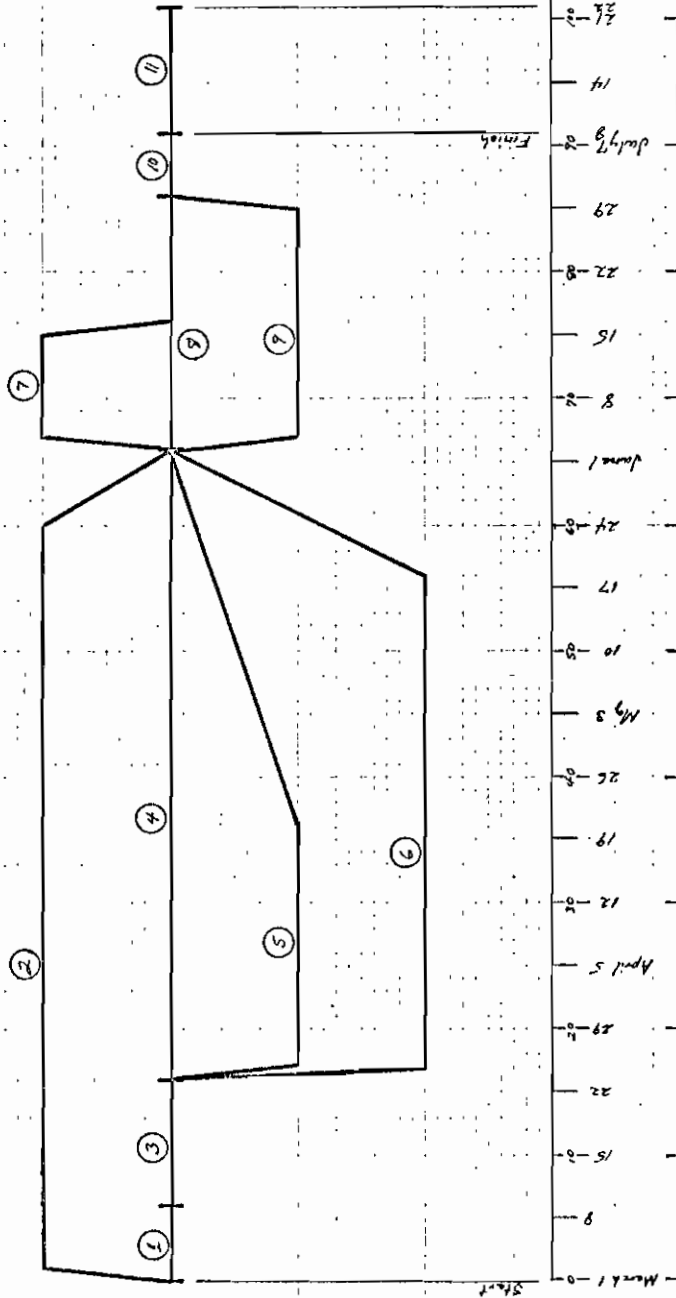
cc: T. G. Melrose

Uranium Pilot Plant Costs

	<u>Man-Hours</u>	<u>Rate</u>	<u>Cost</u>
Technical Direction	100	\$30/hr	\$ 3,000
Engineering	720	\$20/hr	14,400
Technician	920	\$15/hr	<u>13,800</u>
		Total	\$31,200
		Purchased Equipment (including ion exchange resin)	<u>63,300</u>
		Total	\$94,500
		10% Profit	<u>9,500</u>
		Cost Plus Bid	104,000
		Contingency	<u>16,000</u>
		Bottom Line Bid	\$120,000

Above bids are good for 30 days.

PERT Diagram for Construction of Nu-Beth Project Uranium Recovery Pilot Plant



Notes: A day is a work day.

"Finish" means an assembled plant with equipment checked ready for partial disassembly and shipment.



MIDVALE OFFICE

ROCKY MOUNTAIN GEOCHEMICAL CORP.

P. O. BOX 337 • 1323 W. 7900 SOUTH • MIDVALE, UTAH 84047 • PHONE: (801) 255-3558

Certificate of Analysis

Page 1 of 3

Date: February 15, 1977
Client: Nu-Beth
P. O. Box 337
Midvale, Utah 84047
Attn: Grant Buma

RMGC Numbers:

Local Job No.: 76-33-28-SL

Foreign Job No.:

Invoice No.: M 90154

Client Order No.: none

Report On: 41 Samples

Submitted by: Grant Buma

Date Received: 1/16/77

Analysis: Na_2CO_3 , NaHCO_3 , and Chlorine

Analytical Methods: Na_2CO_3 and NaHCO_3 determined titrametrically.
Chlorine determined by specific ion electrode.

Remarks:

cc: enc.
file (2)
LRR/lw

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND = None Detected 1 ppm = 0.0001% 1 Tray oz./ton = 34.286 ppm 1 ppm = 0.0292 Tray oz./tr

<u>Sample No.</u>	<u>g/l</u> <u>Na₂CO₃</u>	<u>g/l</u> <u>NaHCO₃</u>	<u>ppm</u> <u>Chlorine</u>
1A-1	0.11	0.71	17.7
2	0.11	0.84	19.9
3	nil	1.01	19.0
4	0.11	0.88	19.9
5	0.11	0.88	19.9
2A-1	0.42	1.68	28.4
2	0.32	1.85	27.1
3	0.32	1.85	34.6
4	0.32	1.85	26.2
5	0.32	1.93	26.2
3A-1	1.06	2.60	26.2
2	1.06	2.60	27.1
3	1.06	2.60	28.0
4	1.17	2.44	27.1
5	1.06	2.60	24.4
4A-1	1.06	2.60	32.0
2	1.06	2.60	28.4
3			
4	1.06	2.60	28.0
5	1.06	2.77	28.4
1B-1	ND	0.50	21.5
2	ND	0.76	20.0
3	ND	0.67	20.4
4	ND	0.59	20.0
5	ND	0.63	20.0



<u>Sample No.</u>	<u>g/l</u> <u>Na₂CO₃</u>	<u>g/l</u> <u>NaHCO₃</u>	<u>ppm</u> <u>Chlorine</u>
2B-1	ND	0.42	19.5
2	ND	0.42	19.5
3	ND	0.42	19.5
4	ND	0.42	19.5
5	ND	0.42	20.0
3B-1	ND	0.39	18.2
2	ND	0.40	17.3
3	ND	0.39	16.7
4	ND	0.34	16.7
5	ND	0.35	16.7
4B-1	ND	0.30	17.3
2	ND	0.34	16.7
3	ND	0.39	16.7
4	ND	0.40	16.2
5	ND	0.39	16.2
Heads	.032	.75	21.3
formation +2 g/l CO ₃			19.5

By Lawrence R. Reid
Lawrence R. Reid



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA



MIDVALE OFFICE

ROCKY MOUNTAIN GEOCHEMICAL CORP.

P. O. BOX 337 • 1323 W. 7900 SOUTH • MIDVALE, UTAH 84047 • PHONE: (801) 255-3558

Certificate of Analysis

Page 1 of 5

Date: February 14, 1977
Client: Nu Beth
P. O. Box 337
Midvale, Utah 84047
Attn: Grant Buma

RMGC Numbers:
Local Job No.: 76-33-28-SL
Foreign Job No.:
Invoice No.: M 90151

Client Order No.:

Report On: 71 Samples

Submitted by: Grant Buma

Date Received: 1/18/77

Analysis: Na_2CO_3 , NaHCO_3 , Sulfur, Fluorine, Iron, Manganese, Cobalt, Magnesium, Sodium, Potassium, V_2O_5 , Selenium and P_2O_5

Analytical Methods: Na_2CO_3 and NaHCO_3 determined titrimetrically, Sulfur determined leco induction furnace, Fluorine determined by specific ion electrode, P_2O_5 determined colorimetrically, other remaining elements determined by atomic absorption

Remarks:

cc: enc.
file (2)
LRR/lw

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND = None Detected 1 ppm = 0.0001% 1 Troy oz./ton = 34.286 ppm 1 ppm = 0.0292 Troy oz./ton

<u>Sample No.</u>	<u>% Cobalt</u>	<u>% Magnesium</u>	<u>% Sodium</u>	<u>% Potassium</u>	<u>% Iron</u>
tailings 1A	.17	.35	.51	1.50	.90
2A	.84	.33	.54	1.50	.82
3A	.97	.34	.56	1.55	1.00
4A	.96	.34	.57	1.60	.88
1B	1.15	.36	.50	1.55	1.00
2B	.94	.34	.48	1.50	.94
3B	.86	.35	.49	1.60	.90
tailings 4B	.90	.35	.47	1.55	1.08

<u>Sample No.</u>	<u>% Manganese</u>	<u>ppm V₂O₅</u>	<u>% P₂O₅</u>	<u>ppm Selenium</u>
tailings 1A	.024	10	-.01	-1
2A	.016	20	.02	-1
3A	.019	20	.01	-1
4A	.018	20	.02	-1
1B	.021	30	.01	-1
2B	.017	40	-.01	-1
3B	.017	30	.01	-1
tailings4B	.017	30	.01	-1
tailings 11				-1


ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY UTAH

RENO, NEVADA

TUCSON, ARIZONA

<u>Sample No.</u>	<u>% Na₂CO₃</u>	<u>% NaHCO₃</u>	<u>% Sulfur</u>	<u>% Fluorine</u>
494-496	ND	.22	.084	.033
496-499	.034	.33	.067	.041
499-501	.034	.24	.078	.030
501-503	ND	.18	.058	.032
503-505	ND	.17	.152	.030
505-507	ND	.19	.126	.035
508-510	ND	.19	.174	.027
510-512	ND	.27	.248	.040
512-514	ND	.32	.128	.030
514-516	ND	.39	.122	.021
516-518	.009	.30	.106	.023
519-521	ND	.28	.282	.027
521-523	ND	.30	.100	.023
523-525	ND	.20	.074	.095
525-527	.009	.21	.152	.058
527-529	.013	.20	.090	.034
529-531	.009	.22	.258	.039
531-533	ND	.21	.096	.033
533-536	ND	.19	.090	.040
536-538	ND	.25	.103	.042
538-540	ND	.16	.038	.030
541-543	ND	.21	.182	.037
543-545	ND	.19	.056	.036
545-547	ND	.22	.100	.034
547-549	ND	.09	.046	.016

**ROCKY MOUNTAIN GEOCHEMICAL CORP.**

SALT LAKE CITY UTAH

RENO NEVADA

TUCSON ARIZONA

<u>Sample No.</u>	<u>% Na₂CO₃</u>	<u>% NaHCO₃</u>	<u>% Sulfur</u>	<u>% Fluorine</u>
549-551	ND	.09	.086	.012
551-553	ND	.08	.066	.017
553-555	ND	.09	.050	.013
555-557	ND	.10	.104	.022
557-559	ND	.09	.064	.031
559-561	ND	.09	.070	.041
561-563	ND	.07	.120	.024
563-565	ND	.19	.600	.069
565-567	ND	.36	1.04	.073
567-569	ND	.38	.800	.071
569-570	ND	.33	.470	.077
406-407	ND	.17	.018	.030
407-409	ND	.18	.016	.022
409-411	ND	.15	.016	.026
411-413	ND	.15	.026	.022
413-415	ND	.14	.020	.015
415-417	ND	.29	.064	.048
417-419	.017	.36	.334	.117
419-421	.013	.32	.148	.054
421-423	ND	.20	.100	.042
423-425	ND	.19	.052	.035
425-427	ND	.18	.046	.043
427-429	ND	.19	.036	.036
429-433	ND	.22	.352	.040
433-435	ND	.28	.118	.051

-

**ROCKY MOUNTAIN GEOCHEMICAL CORP.**


SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA

Sample No.	% <u>Na₂CO₃</u>	% <u>NaHCO₃</u>	% <u>Sulfur</u>	% <u>Fluorine</u>
435-437	ND	.24	.066	.046
437-439	ND	.22	.043	.039
439-441	ND	.26	.068	.051
441-443	ND	.21	.025	.140
443-445	ND	.31	.040	.041
445-447	ND	.28	.077	.047
447-448	ND	.34	.091	.056
448-450	ND	.19	.022	.028
450-452	ND	.20	.029	.055
452-454	ND	.20	.143	.054
454-455	ND	.21	.056	.073
Carbonate FJR A	ND	.31	.060	.053
Acid FJR B	ND	.31	.055	.063
Tails 1 A	ND	.24	.069	.073
2 A	ND	.26	.069	.041
3 A	ND	.30	.135	.043
4 A	.017	.28	.058	.044
1 B	ND	.21	.111	.042
2 B	ND	.20	.094	.027
3 B	ND	.19	.055	.041
Tails 4 B	ND	.15	.143	.033

By


Lawrence R. Reid**ROCKY MOUNTAIN GEOCHEMICAL CORP.**

SALT LAKE CITY UTAH

RENO NEVADA

TUCSON ARIZONA

Byron Thomas
Rocky Mountain Geochemical
Corp.
1323 West 7900 South
West Jordan, Utah 84084HUFFMAN LABORATORIES, INC.
3830 HIGH COURT, P. O. BOX 350
WHEAT RIDGE, COLORADO 80033
TELEPHONE 424-3232 (AREA CODE 303)Lab. No. 2277701
Date Rec. FEB 8 - 1977
Date Rept. 2/17/77ANALYTICAL REPORT

Lab. No.	Sample No.	Values Found %	
		Total Carbon	Carbonate Carbon
2277701	Carbonate FJR-A	0.49	0.19
2277702	Acid FJR-B	0.48	0.16
2277703	Tailings 1A	0.48	0.26
2277704	Tailings 2A	0.42	0.18
2277705	Tailings 3A	0.41	0.19
2277706	Tailings 4A	0.47	0.15
2277707	Tailings 1B	0.51	0.19
2277708	Tailings 2B	0.50	0.20
2277709	Tailings 3B	0.44	0.11
2277710	Tailings 4B	0.43	0.13
2277711	406-407	0.11	0.09
2277712	411-413	0.10	0.03
2277713	417-419	1.38	0.83
2277714	423-425	0.53	0.46
2277715	429-433	0.48	0.17
2277716	437-439	0.47	0.28
2277717	443-445	1.08	0.57
2277718	448-450	0.27	0.17
2277719	454-455	0.51	0.38
2277720	494-496	0.34	0.04
2277721	501-503	0.21	0.04
2277722	508-510	0.19	0.03
2277723	514-516	2.59	0.99
2277724	521-523	0.84	0.53
2277725	527-529	0.16	0.05
2277726	533-536	0.20	0.17
2277727	541-543	0.40	0.22
2277728	547-549	0.06	0.02

Byron Thomas

HUFFMAN LABORATORIES, INC.

3830 HIGH COURT, P. O. Box 350

WHEAT RIDGE, COLORADO 80033

TELEPHONE 424-3232 (AREA CODE 303)

Lab. No. 2277701

ANALYTICAL REPORT (Continued)

Lab. No.	Sample No.	Values Found %	
		Total Carbon	Carbonate Carbon
2277729	DOH 821-R 553-555	0.28	0.02
2277730	559-561	0.32	0.01
2277731	565-567	1.65	0.98



ROCKY MOUNTAIN GEOCHEMICAL CORP.

MAILING ADDRESS:
P. O. BOX 337
MIDVALE, UTAH 84047

BUSINESS LOCATION:
1323 WEST 7900 SOUTH
WEST JORDAN, UTAH 84084
PHONE: (801) 255-3558

February 24, 1977

Mr. Albert F. Stoick
Nuclear Dynamics, Inc.
200 South Lowell
Casper, Wyoming 82601

Dear Mr. Stoick:

Please find enclosed herewith, Analytical reports charged on Invoice no's. M 90151, 90154 and 90193, forwarded to you February 22, 1977.

Very truly yours,

ROCKY MOUNTAIN GEOCHEMICAL CORP

Grant Buma, President

GB/lw

enc.

NU-BETH

Heavy Mineral Separations

DDH 758 R

Heavy Liquid: Diiodomethane CH_2I_2

Sample Weight: 100 grams

Interval	Wt of Heavy Minerals (grams) (%)
406 - 410	.001
410 - 414	.044
414 - 418	.006
418 - 422	.004
422 - 426	.004
426 - 430	.003
432 - 436	.004
436 - 439.8	.003
439.8 - 444	.006
444 - 448	.008
449 - 455	.015

Heavy minerals included primarily Pyrite with lesser amounts of Magnetite, Barite, and Zircon.

February 3, 1977

G.L. Brooke
File

SUMMARY OF WELL COMPLETION TECHNIQUES

Information on well completion techniques has been received from many sources over the past several months. Each expert has a different idea about how it should be done, and a very active debate will probably continue for years. However, we have reached the point in time when hard decisions will have to be made, if the 5-spot test is to proceed on schedule.

I would like to propose the following method and will set it down on paper. So that, if there is any disagreement we can discuss it fully before the program begins.

First, I think we should use Yelomine casing. It seems to have received the widest acceptance recently, throughout in-situ projects, at least in Wyoming. There may be equally good, less expensive, pipe available, but experimentation with the lesser known casings should be left for sometime in the future.

In general I think we should follow W. H. Elliott's procedure for well completion, with a few minor exceptions.

The step by step procedure should be as follows:

1. Drill the pilot hole to the coring depth.
2. Ream the pilot hole to 9" diameter.
3. Run 5" Yelomine casing into the hole, with the "modified bottom section" and cap attached. Place centralizers at 100' intervals, as the casing goes in the hole.
4. Flush the annular space with fresh water.
5. Cement the entire annular space, forcing cement back out the surface, to insure a complete cementing job.

A well head for cementing (attached) has been designed by Cleveland Cliffs. This head allows the cement to be forced to the bottom of the hole by a Halliburton plug and a column of water. The plug is held in place by set screws, etc., until the cement is pumped into the hole. It is then released and pressured to the bottom. Cliff's have found that the plug is easier to drill out if it is followed with about 10 feet of cement and then with the water. Cementing of the plug keeps the plug from moving around when it is attacked by the drill bit.

The Halliburton cementing tables suggest the following quantities of cement for a 9" bore hole and 5.5" O.D. casing - 2.0706 gallons per linear foot. The same tables estimate that .9997 gallons of water, per linear foot, will be required to force the cement and plug to the bottom in a casing with an I.D. of 4.95".

The cement mixture should be essentially as proposed by Elliott, minus the fly ash. In view of the difficulty in getting the fly ash to the project site, and getting a commercial cement mixer, when needed, I suggest that we mix the cement in two 500 gallon stock tanks on location.

The following quantities of materials are needed for a 500 foot, 9" hole, containing a 5.5" I.D. casing:

100 sacks (9,400 lbs.) Type II cement
550 lbs. (5.9%) Gel
450 lbs. (4.8%) Coarse Mica
910 gals. (9.1 gal/sk) Water

Calcium chloride can be added, but should not exceed 2%.

The Halliburton tables recommend 6.3 gals/sk of quick drying cement. The amount of water can be varied as conditions dictate.

A small amount of the cement mixture should be save in an ice cream carton, etc., for use in determining setting-up characteristics.

The casing will have to be tied-down, or weighted at the collar, until setting-up takes place. This process will likely take a minimum of 24 hours, and 36 to 48 hours, or longer, is recommended.

6. Drill out the cement remaining in the casing, plus the Halliburton plug, and the bottom cap.
7. Core the mineralized interval, and flush with fresh water. If the cuttings are difficult to remove from the hole, some Baroid Quik Foam may be added to the circulating medium.
8. Hang well screen in 4 7/8" (cored interval) bottom section. Some suggest a 2" stainless steel Johnson screen, others say a 4" can be placed in this size hole. A good compromise would seem to be a 3 5/8" O.D., 3" I.D. Johnson screen, with a 5" stainless steel, self sealing packer. This packer will have to be reworked to fit snugly into the 4.911" (minimum) I.D. Yelomine casing.

Other:

The pump for the pumping test and in the recovery hole will have to be placed in the casing, somewhere above the screen.

If it is determined that coring cannot be carried on through the casing without excessive damage to the Yelomine pipe, then I would suggest that the coring be done before the casing is set. A wooden

plug could be placed at the top of the cored interval, so that no cement could enter the mineralized part of the hole.

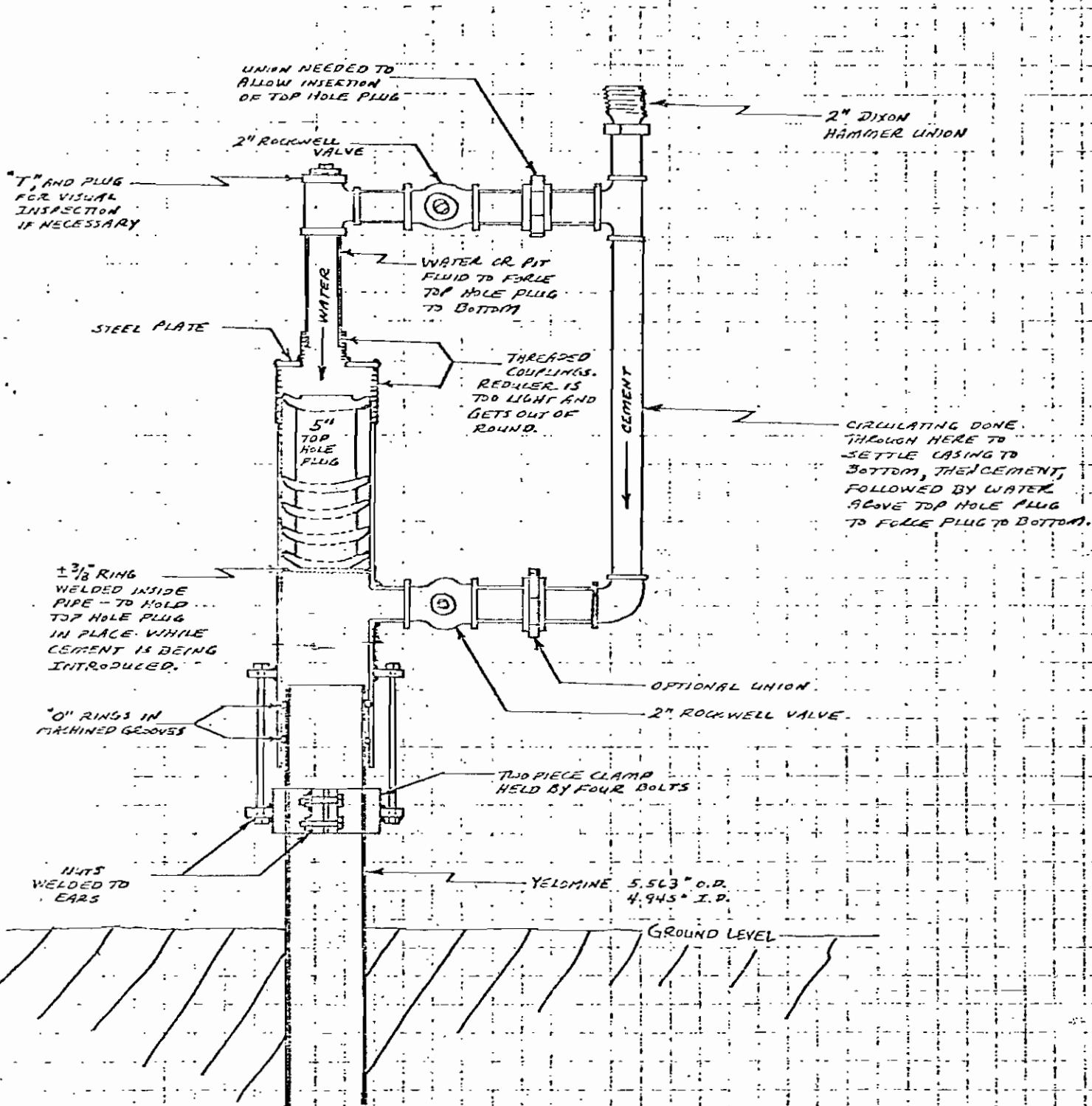
GLB:mkp

cc: G. Biemesderfer
T. Melrose
A. Stoick

11/22/76 - CKR

WELL HEAD FOR CEMENTING

APPROXIMATE SCALE: 1" = 10"





ROCKY MOUNTAIN GEOCHEMICAL CORP.

MAILING ADDRESS:
P. O. BOX 7366
SPOKANE, WASHINGTON

LABORATORY ADDRESS:
N. 6319 HELENA
SPOKANE, WASHINGTON
PHONE: 489-3310

January 27, 1977

Mr. Albert F. Stoick
Nuclear Dynamics, Inc.
200 South Lowell
Casper, Wyoming 82601

Dear Al,

Please find enclosed, 2 copies of histograms of trace element analyses on holes 829 & 758. This data is not complete, but I am sending this to you for your information. When the remainder of the analyses are completed I shall have them plotted and send them to you also along with an evaluation. I have postponed that work in order to concentrate on the design of your IX plant in accordance with instructions from Tom. I shall, however, come back to this work as time permits and provide you with a final report on the feasibility study and an evaluation of the core data and recommendations for future work.

We did rerun the samples in the region of 446-447.5 on hole 758 and our values are still much higher than Skyline's:

446-447 - 480 ppm

447-447.5 - 620 ppm

I must conclude that the samples have been misplaced in the core box before we received them and after submission to Skyline. The rest of the numbers check out quite well.

I'll keep you posted on our progress in designing the plant. If you have any questions, please don't hesitate to call.

Best regards,
ROCKY MOUNTAIN GEOCHEMICAL CORP.

Grant Buma, President

GB/nb

NU-BETH

Heavy Mineral Separations

DDH 758 R

Heavy Liquid: Diiodomethane CH_2I_2

Sample Weight: 100 grams

Interval	Wt of Heavy Minerals (grams) (%)
406 - 410	.001
410 - 414	.044
414 - 418	.006
418 - 422	.004
422 - 426	.004
426 - 430	.003
432 - 436	.004
436 - 439.8	.003
439.8 - 444	.006
444 - 448	.008
449 - 455	.015

Heavy minerals included primarily Pyrite with lesser amounts of Magnetite, Barite, and Zircon.

Drill Hole No.	<u>SP756 R</u>	Collar Elev.	<u> </u>	Coord.	<u> </u>	Depth	<u> </u>
Location	<u> </u>	Cored Interval	<u> </u>	Cored Recovery	<u> </u>		

Logged By: BUMA
Date 3/16/77

[illegible]

Drill Hole No. SP 758 R
Location _____

Collar Elev.

Coord.

Depth

Logged By:
Date

Cored Interval
Cored Recovery
BLM A
3116177

[illegible]

Date 3/16/77[illegible]

Drill Hole No. SP 756 R
Location _____

Collar Elev.

Coord.

Depth

Logged By:
Date

Date 3/17/77

Cored Interval
Cored Recovery

Cored Interval
Cored Recovery

Rock Type		Depth and Formation		Lithology										Alteration				Remarks
				Grain Size	Sorting	Cementation	Mineral and Rock Fragments			Organic Matter	Porosity	Color	Intensity	Oxidized	Reduced	R/A C.P.S.		
				M-F	POOR	POOR	60	35	4	-	1%	HIGH	LIGHT OLIVE GRAY	0		/	-	DRY CORE
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"
				"	"	"	60	35	4	-	1%	"	"	0		/	-	"



MIDVALE OFFICE

ROCKY MOUNTAIN GEOCHEMICAL CORP.

P. O. BOX 337 • 1323 W. 7900 SOUTH • MIDVALE, UTAH 84047 • PHONE: (801) 255-3558

Certificate of Analysis

Page 1 of 4

Date: January 13, 1977
Client: NU-BETH
P.O. Box 337
Midvale, UT 84047
attn: Grant Buma

RMGC Numbers:

Local Job No.: 76-33-28SL

Foreign Job No.:

Invoice No.: M 90048

Client Order No.:

Report On: 63 samples

Submitted by: Mr. Buma

Date Received:

Analysis: Iron, Manganese, Calcium, Magnesium, Sodium, Potassium

Analytical Methods: Determined by atomic absorption.

Remarks:

cc:
enc.
file (2)

LRR/bar

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND = None Detected 1 ppm = 0.0001% 1 Troy oz./ton = 34.286 ppm 1 ppm = 0.0292 Troy oz./ton

Sample No.	% Fe	% Ca	% Mg	% K	% Na
DDH 829R					
494-496	1.0	.25	.37	1.7	.76
496-499	1.1	3.00	.41	1.5	.70
499-501	.9	3.20	.33	1.4	.64
501-503	.7	.29	.34	1.6	.69
503-505	.8	.25	.35	1.8	.73
505-507	.7	.23	.34	1.7	.70
508-510	.8	.26	.33	1.5	.67
510-512	1.5	.36	.52	1.7	.81
512-514	.7	.26	.30	1.4	.60
514-516	.7	5.5	.29	1.3	.61
516-518	.7	.50	.31	1.4	.64
519-521	.8	.36	.26	1.3	.63
521-523	.6	2.55	.28	1.4	.60
523-525	.7	.30	.34	1.5	.72
525-527	1.0	.52	.44	1.6	.77
527-529	.8	.47	.40	1.5	.75
529-531	.8	.56	.45	1.6	.76
531-533	.7	.60	.40	1.6	.78
533-536	.7	.62	.43	1.6	.72
536-538	.8	.61	.45	1.8	.79
538-540	.6	.43	.35	1.8	.75
541-543	.8	.60	.57	1.9	.81
543-545	.7	.59	.46	1.8	.80
545-547	1.0	.51	.46	1.7	.79
547-549	.4	.14	.20	1.4	.70



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA

<u>Sample No.</u>	<u>% Fe</u>	<u>% Ca</u>	<u>% Mg</u>	<u>% K</u>	<u>% Na</u>
<u>DDH 829R</u>					
549-551	.5	.24	.22	1.4	.66
551-553	.5	.17	.22	1.4	.68
553-555	.5	.18	.24	1.5	.71
555-557	.5	.20	.27	1.7	.72
557-559	.5	.19	.25	1.5	.68
559-561	.6	.19	.25	1.7	.71
561-563	.5	.15	.23	1.6	.74
563-565	2.5	.42	.95	2.2	1.18
565-567	3.4	4.40	1.50	2.0	1.00
567-569	3.0	2.50	1.85	2.3	1.16
569-570	2.3	2.35	1.95	2.1	1.17
<u>DDH 758R</u>					
406-407	.8	.23	.28	1.3	.58
407-409	.6	.23	.29	1.4	.70
409-411	.8	.22	.28	1.5	.66
411-413	.7	.38	.27	1.0	.56
413-415	1.2	.32	.25	1.1	.54
415-417	1.4	1.80	1.40	1.7	.91
417-419	1.9	1.75	1.55	1.7	.97
419-421	1.0	1.00	.80	1.8	.85
421-423	.8	.90	.70	1.9	.83
423-425	1.0	.95	.75	1.9	.82
425-427	.7	.75	.60	1.7	.81
427-429	.7	.60	.50	1.9	.81
429-433	1.3	.65	.55	1.7	.80
433-435	1.4	1.20	1.05	1.9	.95



<u>Sample No.</u>	<u>% Fe</u>	<u>% Ca</u>	<u>% Mg</u>	<u>% K</u>	<u>% Na</u>
DDH 758R					
435-437	.9	.80	.65	1.9	.89
437-439	.9	.70	.57	1.8	.82
439-441	1.2	1.00	.80	1.9	.88
441-443	1.1	1.15	.85	1.8	.87
443-445	1.9	1.80	1.45	1.9	1.00
445-447	1.6	1.45	1.20	1.9	.95
447-448	1.5	.50	.60	1.7	.94
448-450	.9	.50	.50	1.6	.82
450-452	.9	.65	.55	1.7	.84
452-454	1.6	.85	.70	1.6	.89
454-455	.9	.95	.75	1.8	.92

(DDH 758R)

						<u>% MN</u>
Carbonate FJR-A	.8	.60	.34	1.4	.59	.018
Acid FJR-B	.7	.50	.33	1.5	.56	.014



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA

By

Lawrence R. Reid



MIDVALE OFFICE

ROCKY MOUNTAIN GEOCHEMICAL CORP.

P. O. BOX 337 • 1323 W. 7900 SOUTH • MIDVALE, UTAH 84047 • PHONE: (801) 255-3558

Certificate of Analysis

Page 1 of 4

Date: January 31, 1977
Client: NU-BETH
P.O. Box 337
Midvale, UT 84047
attn: Grant Buma

RMGC Numbers:
Local Job No.: 76-33-28 SL
Foreign Job No.:
Invoice No.: M 90106

Client Order No.: none

Report On: 63 samples

Submitted by: Mr. Buma

Date Received: 1/7/77

Analysis: Al_2O_3 and SiO_2

Analytical Methods: Determined by atomic absorption.

Remarks:

cc: enc.
file (2)

LRR/bar

All values are reported in parts per million unless specified otherwise. A minus sign (—) is to be read "less than" and a plus sign (+) "greater than." Values in parenthesis are estimates. This analytical report is the confidential property of the above mentioned client and for the protection of this client and ourselves we reserve the right to forbid publication or reproduction of this report or any part thereof without written permission.
ND = None Detected 1 ppm = 0.0001% 1 Troy oz./ton = 34.286 ppm 1 ppm = 0.0292 Troy oz./ton

<u>Sample No.</u>	<u>% Al₂O₃</u>	<u>% SiO₂</u>
DDH 829K		
494-496	8.9	81.6
496-499	9.8	70.5
499-501	7.4	76.8
501-503	7.7	83.1
503-505	7.3	83.7
505-507	8.0	81.9
508-510	7.2	84.0
510-512	11.8	76.5
512-514	6.5	77.7
514-516	6.3	71.1
516-518	6.7	83.4
519-521	6.0	83.7
521-523	6.0	79.8
523-525	7.3	84.0
525-527	7.6	81.0
527-529	7.6	82.5
529-531	7.3	81.0
531-533	7.2	81.6
533-536	7.5	81.9
536-538	8.9	79.2
538-540	7.6	83.4
541-543	8.0	79.5
543-545	7.8	81.9
545-547	7.8	82.2
547-549	5.8	86.1



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA

Sample No.	% Al_2O_3	% SiO_2
DDH 829K		
549-551	6.1	84.3
551-553	6.1	86.7
553-555	6.6	84.6
555-557	6.9	84.3
557-559	6.4	84.3
559-561	6.8	83.4
561-563	6.5	84.9
563-565	12.5	72.6
565-567	12.1	60.6
567-569	13.6	61.8
569-570	12.7	64.2
755R 406-407	6.8	85.2
407-409	7.0	84.9
409-411	7.0	85.2
411-413	4.3	88.5
413-415	5.2	85.2
415-417	10.1	73.5
417-419	11.6	70.2
419-421	9.5	75.6
421-423	8.1	80.1
423-425	8.1	81.3
425-427	7.8	79.2
427-429	7.8	82.5
429-433	7.7	81.0
433-435	9.5	75.0



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY, UTAH

RENO, NEVADA

TUCSON, ARIZONA

Sample No.	% Al_2O_3	% SiO_2
435-437	7.9	80.7
437-439	8.0	78.6
439-441	8.8	76.2
441-443	8.1	78.0
443-445	11.0	69.3
445-447	9.4	74.7
447-448	9.8	78.0
448-450	7.9	80.4
450-452	7.5	79.2
452-454	8.2	78.0
454-455	8.2	78.3
FJR-A	7.5	81.0
FJR-B	7.6	82.2



ROCKY MOUNTAIN GEOCHEMICAL CORP.

SALT LAKE CITY UTAH

RENO, NEVADA

TUCSON, ARIZONA

By Lawrence R. Reid
Lawrence R. Reid