5.0 BASELINE FORECAST

Demand and production forecasts are given below for the years 1983 and 1990. These estimates were derived from a combination of publicly available forecasts. The projections presented here were done in this manner because historically there have been variations between individual forecasts. Taking an average of the various estimates gives a reasonable and representative forecast without having to determine the merit of each individual forecast and its implicit assumptions. In lieu of developing new forecasting models for demand and production a range and average will be given as a representative forecast, as derived from five published forecasts.

5.1 REACTOR FUEL DEMAND FORECAST

Demand forecasts from five different sources were used to develop a representative forecast. Reactor demand can be met in three ways: (1) U.S. production, (2) foreign production, and (3) inventory reduction. Consumption and reactor demand are assumed to be equivalent in this section. Table 5.1 shows the demand forecasts, their sources, and the representative forecast.

TABLE 5.1. Annual U.S. Uranium Demand in Tons $U_3O_8 \times 10^3$

Source	Year	
	1983	1990
DOE Grand Junction [(a)]	16.5	23.8
Power Magazine [(b)]	14.2	18.8
Rocky Mountain Energy [(c)]	16.3	22.5
Nuclear Assurance Corporation [(d)]	16.7	20.7
Pickard, Lowe, and Garrick [(e)]	15.7	20.5
Representative Forecast		
Average	15.9	21.3
Range	14.2-16.7	18.8-23.8

Sources: (a) DeVergie, 1982. Table 5. (Based on the August 1981 low case prepared by the DOE Office of Uranium Enrichment and Assessment).
(b) Catalano, 1983a. Figure 3.
(c) Lang, 1982. Figure 13.
(d) Leamon, 1982. Figure 5. (U.S. Reasonable Case Demand).
(e) Nuclear Fuel 1983b, p. 8.

5.2 DEMAND/OTHER USES

John Hunter, Chief of Reactor Operations, Department of Energy, Washington, D.C., can provide information on uranium demand/use at non-commercial reactors. This information is not immediately available to the public. Since this segment makes up only a small percentage of total uranium demand, it will not be considered here in the ore production forecast.

5.3 ORE PRODUCTION FORECASTS

The same five sources that were used in Section 5.1 to develop the reactor fuel demand forecast will be utilized in this section to generate a representative ore production forecast. Explicit forecasts of uranium production were made by two of the sources DOE Grand Junction (DeVergie 1982) and Power (Catalano 1983a). For the other three sources, production forecasts must be derived from the demand forecasts.

Rocky Mountain Energy (Lang 1982) estimated inventory usage in each year of their 1982 to 1996 domestic market demand forecast. Combining their estimated inventory usage with an estimate of imports yields a production forecast as a percentage of demand. For 1983 an estimate of 30% imports was used. "Presently, about 28% of all uranium in the U.S. is imported, and this figure is expected to increase" (Catalano, 1983a). For 1990 imports were estimated as 37.5% of domestic demand which is the proposed import limit. The resulting Rocky Mountain Energy uranium production forecasts as a percentage of demand for 1983 and 1990 are approximately 61% and 55%, respectively.

Using the DOE Grand Junction (DeVergie 1982) and Power (Catalano 1983a) forecasts as well as the derived estimate of demand for Rocky Mountain Energy it was determined that U.S. U_3O_8 production is an estimated 66.7% and 54.7% of reactor demand for the years 1983 and 1990 respectively. Using these values, forecasts of U_3O_8 production were made from the Nuclear Assurance Corporation (Leamon 1982) and Pickard, Lowe & Garrick (Nuclear Fuel 1983b) forecasts of reactor demand. Table 5.2 shows the five U_3O_8 production forecasts and the representative forecast. These forecasts consider only uranium demand for commercial reactors, i.e. demand for other uses is not included as mentioned in Section 5.2. In Table 5.3 the U_3O_8 production is converted to ore production using estimated percent average ore grade as obtained from the U.S. DOE.

TABLE 5.2. Annual U.S. Uranium Production in Tons $U_3O_8 \times 10^3$

Source	Year 1983	Year 1990
DOE Group Junction [a]	10.9	9.5
Power Magazine	10.6	7.6
Rocky Mountain Energy [b,d]	9.9	12.4
Nuclear Assurance Corporation [c,d]	11.1	9.4
Pickard, Lowe & Garrick [c,d]	10.5	9.3
Representative Forecast		
Average	10.6	9.6
Range	9.9-11.1	7.6-12.4

Unless otherwise noted, references are the same as Table 5.1.
(a) Figure 8. (40 percent import limit).
(b) Assumes 30% imports for 1983 and 37.5% for 1990.
(c) Assumes 66.7% U.S. reactor demand is supplied by U.S. production for 1983 and 54.7% for 1990. (This is the average of the first three estimates).
(d) Estimate is derived from U.S. reactor demand forecast.

TABLE 5.3. Annual Ore Production

(Year)	(Tons U_3O_8)	Average Ore Grade, % [a]	(Tons Ore)
1983	10,600	0.12	8,833,333
1990	9,600	0.13	7,384,615

(a) Conventionally mined and milled ore only. Information on percent average ore grade obtained from the DOE, Grand Junction Area office, Grand Junction, Colorado.

6.0 RADIONUCLIDE EMISSIONS TO AIR

This section will develop several important aspects of radionuclide emissions to air beginning with a discussion of the parameters and details involved in developing a model for emission measurement. This will be followed by a detailed description of three methods to measure/monitor radionuclide emissions followed by special topics on instrument calibration and vent flow measurement.

As an alternative method for reducing radionuclide emissions, increasing the effective height of release has been considered. A discussion of the increased cost to production resulting from the installization of a 20-meter-high vent stack will be given.

This section will then conclude with a discussion of the impact that current and projected mining activity (ore production) will have on the magnitude of radionuclide emissions to air.

6.1 TECHNIQUES FOR MONITORING ^{222}Rn EMISSIONS FROM UNDERGROUND URANIUM MINES

The determination of ^{222}Rn emissions in uranium mine ventilation exhausts can be reliably made by direct field measurements of radon concentration in the air exhausted from the vent and the vent air-flow rate. It is impractical to rely on measurements of the concentrations of radon or its short-lived daughters taken underground to determine radon emission rates because the underground measurements are normally taken only in areas where active mine work is in progress. Those areas usually receive the main flow of ventilation air and therefore, with the exception of lunch and shop areas have some of the lowest radon and radon daughter concentrations in the mine. However, a large portion of the radon emanating surfaces of most mines consist of old mined-out areas. Those areas are often bulkheaded (closed off) from the active areas of the mine and receive only enough ventilation to keep them at a negative pressure relative to the active areas. Since a large part of the inactive area is often inaccessible, there will be no radon or radon daughter measurements available from them. The air pumped from those areas to the surface tends to have relatively high ^{222}Rn concentrations but low flow rates and usually contains a large fraction of the total emission of radon.

A model for predicting future vent emissions from past radon emission rate measurements or estimating them from other kinds of measurements, has not been completed. Jackson et al. (1979 and 1980) found that on average, about 50% of the variation of the radon emission from uranium mine vents could be related to the cumulative production of the mine (tons of ore) since it was first opened. This correlation was predicted on the theory that a mine expands in volume during its life, creating additional radon emanating surfaces. The radon emanation from the surfaces containing exposed ore should be greater per unit area than that from worked-out surfaces where the ore has been extracted. However, there is always a residual quantity of uranium minerals in the worked-out surfaces since there is a limit to the ore grade which can economically be mined. As the mine ages, the worked-out areas continue to expand while the surfaces being mined at any given time remain relatively constant. Thus, eventually the worked-out areas grow so large that the emanated radon from those areas exceeds that from exposed ore and the radon emanation will increase with some functional relationship to the total amount of ore extracted. If the mine is considered to be a lengthening tube of constant cross section, then a linear relationship is predicted.

This partial model does not consider several factors which can be extremely important in making predictions for individual mines or vents. Perhaps the most important of these is the stability of the rock which is being mined. After a section of a mine has been abandoned, if the rock is unstable there will be many rock falls and cave-ins after timbers rot and pillars have been removed. Whether these cave-ins result in a greater or lesser amount of radon emanation will depend on whether there is a greatly increased surface area exposed or whether that area of the mine becomes totally sealed from ventilation to the outdoors. Other important factors that could modify the model are the degree to which water is transporting radon to the various surfaces, and the average porosity and permeability of the rock structures in individual mines or ventilated areas.

Thus, while it may be possible to relate radon emissions to production on an industry-wide average, it is not reliable to use this existing partial model to predict emissions from a specific mine for which no measurements are available. The accuracy of the model for extrapolating to future emissions from mines where measurements have been taken in the past, will depend on how closely its assumptions fit the practices in individual cases, and on the perturbations from rock fall already mentioned.

For the purposes of other sections of this report, the partial model will be used to predict increases from previous mine measurements as the mine ages. For those mines which have been reported to be totally shut down with the vents sealed, it will be assumed that there are no current or future emissions. For mines where only a small fraction of the mine near the shaft is occasionally ventilated to permit access to water pumps and the rest of the mine is sealed and bulkheaded from the ventilated area, no calculations will be made. Emissions from such mines may be substantial for short intervals of time when vent fans are turned on, but the average emissions should be only a small fraction of the total measured previously for the entire mine. In this report, these mines will be classified as sealed. Based on these assumptions, the effect of mine shut down and capping will be to totally remove emissions from a fraction of the mines where measurements have been taken. Conversely, increases in production will increase emissions at the remaining mines.

6.1.1 Radon Measurements

There are three potential approaches for monitoring the quantity of ^{222}Rn emitted from uranium mine vents in a selected time interval. In each of these approaches, one measures an average concentration of ^{222}Rn which must be multiplied by the measured air-flow rate of the vent in order to estimate the quantity per unit time. The three radon measurement techniques are: grab sampling,, continuous sequential sampling, and integrating sampling. Certain elements of these sampling techniques are characteristic of stack monitoring for any species and not merely for radon or other radionuclides.

Grab sampling is, as the name implies, the collection of a sample of air from the stack during a single, short period of time or several samples during sequential, short periods of time. It is simple, direct, and normally low cost in terms of equipment. It does, however, pose problems as to the representativeness of the sample(s), and tends to be a labor-intensive technique in terms of manhours per data point.

Continuous sequential sampling is typified by an instrument that is installed at the stack to continuously draw a fraction of the exhaust air and pass the air to an analyzer. The analyzer monitors the concentration of some constituent, records the measurement at intervals, and resets itself to collect and record a new measurement. The primary use of this type of instrumentation is to monitor rapidly changing concentrations. Continuous samplers can establish their time dependency and the range of concentrations which may be encountered when grab sampling. Of course, data taken over an extended interval(s) can also be averaged. Since the instruments can give enough measurements to establish both the average and the dispersion of concentrations, they are useful for research applications. The instruments can provide for continuous monitoring of stacks where grab sampling is not satisfactory because of large and unpredictable fluctuations in concentrations.

One drawback of the automatic sequential sampling method is that the equipment is complex and costly. Calibrations and quality assurance procedures for this technique generally require more careful testing of secondary effects on the efficiency from such characteristics of the instrument as the rate of flow, pressure differentials, and air leaks. Other characteristics that can affect measurements are the instrument's sensitivity to ambient temperature and to constituents of the ambient air such as moisture, and its response to particulates in the airstreams. This type of instrument can be susceptible to damage in hostile environments. Calibrations can change with time, necessitating periodic maintenance and frequent operating checks. Costs can be substantial, but since large amounts of data are collected, normally the unit cost per data point is small.

Integrating samplers continuously collect a portion of the total stack flow or just a portion of the constituent being measured over an extended interval. Then the samplers are normally removed from the stack and the total content of the desired constituent is measured. These devices are useful for determining the average concentration with a sampling interval which may range from hours or days up to a year or more depending on the individual instrument.

Some integrating samplers are designed to function without moving parts. For example, the species of interest may enter an evacuated chamber through a small orifice or a particulate species may be collected electrostatically as long as a sufficient electrostatic field can be maintained. Radionuclide exposure rates can be integrated from the darkening of tracks or the increasing number of tracks created in radiation sensitive materials. An example of a mechanical integrator is an air filter that can sample continuously until the particulate loading causes a significant loss of flow rate which causes the sampling to cease. Non-mechanical integrating samplers are often simple in design and low in unit cost. These non-mechanical types are useful when large numbers of long-term measurements are desired. However, only a great deal of study can establish that there is a consistent response-to-concentration relationship for integrators. Nonlinear response characteristics, if present, can mean that the average recorded for the sampled interval is incorrect when a period of high concentration and one of low concentration occurs in that interval. Effects from detector saturation, fading of the response characteristic, background noise variation during long intervals, and changes in the flow rate through the detector as well as the stack must be considered and if necessary, compensated for. Sensitivity of the detector to other constituents which might be present must also be considered. While many of these problems are not unique to integrators, the nature of the measurement techniques can make it difficult to detect such problems from an examination of the data. However, when properly applied, these integrators can provide a large number of economical long-term measurements. The application of these three approaches to radon measurement and some of the equipment typically used will be covered in the next sections.

Grab Sampling and Radon Measurement

The most common technique for ^{222}Rn measurement is the scintillation flask grab sample (device manufactured by companies No. 2, 4, 5, 6 listed in Table 6.1). In this technique, one uses a flask which has been internally coated with a scintillator, such as zinc sulfide. It is filled with the

TABLE 6.1. Manufacturers of Radon Analysis Equipment

1. Aero Vironment
   145 Vista Avenue
   Pasadena, California

2. Eberline Instrument Corporation
   P.O. Box 2108
   Santa Fe, New Mexico 87501

3. EDA Instruments, Incorporated
   5151 Ward Road
   Wheat Ridge, Colorado 80033

4. Johnston Laboratories, Incorporated
   3 Industry Lane
   Corkeysville, Maryland 21030

5. Pylon Electronic Development Company Ltd.
   47 Colonade Road
   Ottawa, Canada KZE 729

6. Rocky Mountain Scientific Glassblowing Company
   2520 Galena
   Aurora, Colorado 20010

7. Terradex Incorporated
   460 North Wiget Lane
   Walnut Creek, California 94598

atmosphere to be sampled, either by pumping air through a flask designed for flow-through operation or by evacuating a flask and allowing the air being sampled to fill the flask.

In the air filling process, the air is usually filtered at the inlet to establish a point in time when there are no radon progeny present in the flask and to prevent the contamination of its interior with long-lived alpha particle emitting radionuclides. The inlet air for the flask is collected from either an access port in the vent or from a point in the mouth of the vent that has been determined to have a uniform outward air flow pattern. If large quantities of water droplets are present, an inline trap should be used to prevent the entry of water into the inlet filter.

After waiting for about four to five hours to permit ingrowing radon progeny to approach equilibrium with the radon, the scintillation flask is placed on a bare photomultiplier tube. This tube can detect scintillations coming from the interaction of alpha particles with the interior coating. The photomultiplier tube is housed in a light-tight container. To prevent damage to the tube, its operating bias voltage is removed whenever the container is opened. It is good practice to place the scintillation flask in the light-tight container and close it for at least five minutes prior to counting to allow the decay of phosphorescence that is induced in the scintillator and photomultiplier tube during exposure to light. After the delay, the scintillations are integrated for an interval that depends on the radon concentration and the sensitivity and precision desired. For monitoring uranium mine stack concentrations, ten minutes to one hour are usually sufficient to permit a precise measurement. Following the count, the air is evacuated from the flask and the flask is rinsed several times with filtered ambient outdoor air. The flask can be stored either evacuated or filled with filtered outdoor air. After a delay of six or more hours, the flask can be returned to the counting chamber to permit determination of its background for subsequent counts.

It is typical to encounter a memory effect when plastic flasks are used, from the finite solubility of radon in plastic materials. This effect was reported by Jackson et al. (1979) to be about 1/2 to 1% of the previous count. It would be expected to diminish with a half life somewhat shorter than the radioactive half-life of radon because of outgasing. Buildup of static charges on the surfaces of the flask during handling can cause serious measurement errors when using nonmetallic flasks in dry atmospheres. The buildup of charge can influence the location of deposited short-lived radon progeny in the flask and thus the detection efficiency. Care should be used when handling flasks not to wipe their surfaces. Commercial antistatic sprays are helpful, but should be applied at least 4 to 5 hours before making a measurement to allow the decay of previously deposited radon progeny.

As noted previously, results based on grab samples can be difficult to defend if there is no evidence showing the dispersion or range of results that can be expected in a given sampling situation. In particular, radon concentrations in mine air have been shown to have a diurnal cycle caused in turn by the diurnal cycle of the air density. Smaller variations may be caused by the air temperature and from the effects on the underground air density produced by winds blowing across or into portals and vents. In addition, regular mining operations and conditions such as the position of the elevators in shafts or blasting could produce short-term changes in the radon concentration. Profound effects can be expected from fan shutdowns or changes in the underground ventilation air path. Thus, it is always necessary to use a sampling protocol which establishes the dispersion of measurements when grab sampling.

As minimum, duplicate samples collected on two different days at different times of the day can give information about the magnitude of cyclic variations versus replication errors. However, even when this type of sampling is used, it is best to avoid the times of blasting or times when the ventilation of the mine can be expected to be abnormal, if one is attempting to obtain average emission rates from a few measurements. Of course, the abnormal conditions could cause such profound radon concentration changes that they would significantly affect the average. This minimum protocol was followed by Jackson et al. (1980) and was evaluated using two systems of continuous sequential monitors and a statistical evaluation of the dispersion of the measurements. While that interpretation indicated that the relative standard deviation of measurements involved in counting scintillation flasks was about 4% when four samples were averaged, the sampling errors appeared to be larger. The ratio of the maximum observed result to the average result measured hourly for about one week periods at three vents ranged from 1.13 to 2.0. At a fourth vent, variations caused by cyclic vent shutdowns produced ratios as high as 7.3. While Jackson et al. (1980) determined that these variations tended to produce a relatively small error in their measurement of the annual average radon emission from the entire U.S. mining industry, based on grab sample determinations of radon in exhaust air, their effect on the average emission rate determined for a single mine can be substantial. Where possible, a large number of grab samples should be collected so that patterns of emission rates and correlations with operations can be established. However, if a very detailed and extensive study at a few vents is to be performed, the cost of grab sampling could be prohibitive and some form of continuous sequential monitoring should be considered.

When grab sampling, the use of replicates taken at different times, days of the week (operating and holidays), and taken over a substantial portion of the year and analyzed with a statistical technique known as the analysis of variance can yield information about the long- and short-term concentration changes that can be expected. The data-handling methods and the design of experiments used with this technique are covered in standard statistical texts and will not be repeated here. As a basis for estimating the average emissions from any mine vent using grab sampling, the following protocol is suggested.

1. On a work day morning, at least one hour after shift change and blasting, collect a duplicate set of grab samples within a five-minute interval and measure the vent flowrate.

2. Collect a second duplicate set of grab samples on the afternoon of the same day or within a few days of the original set (sample on working days only) and remeasure the vent flowrate. The afternoon sample should be collected at least one hour after lunch break and/or blasting.

3. Collect a set of two samples, one 5 to 15 minutes after blasting and the other 40 to 60 minutes after blasting. These samples will give information about the concentration maximum, and the rate of its decrease following a blast.

4. Collect a set of samples on a weekend or at least two hours after the last working shift on a weekday. (If ventilation changes have been made, see Step 5 below.)

5. If any shutdown of vent hole ventilation fans or if a flow reversal is a normal procedure at the mine, sample all of the vents at the mine within the intervals from 30 to 60 minutes before returning the ventilation fan(s) to normal status, from 10 to 30 minutes after returning the ventilation to normal, and again in the interval from 60 to 120 minutes after the ventilation fans have returned to normal operation. Measure the vent flow rates within 30 minutes of each sampling (with the fans on and with the fans off). Indicate the directions of flow as necessary.

6. Repeat Steps 1 or 2 on quarterly intervals for one year.

This protocol will require at least 14 grab samples per vent, collected over a one-year interval. It should be capable of accurately determining the annual average emission rate of ^{222}Rn for each mine vent and estimating the precision of the measurements. It should give enough information about the effect of operations on the radon emission rates to determine if they are significant enough to require additional study. Since measurements are continued for one year, relationships of radon emission rates to continuing production and excavation should be established.

Continuous Sequential Sampling for the Measurement of ^{222}Rn

There are two and possibly three commercially available continuous sequential sampler designs that could potentially be used for measuring radon in mine ventilation air. The first is an automated flow-through scintillation flask. The Eberline RGM II is an example manufactured by company No. 2 listed in Table 6.1. The Eberline instrument uses a precalibrated microprocessor to compute and print results in units of pCi/ℓ. This type of device draws filtered air continuously through a scintillation flask, stores scintillations detected by its photo-multiplier tube for an interval, computes the result, clears its scaler, and accumulates new counts at the preset intervals. The 1983 price for this unit is approximately $7800. Since the ingrowing radon daughters accumulate on the walls of the detector, their alpha particle emissions contribute to the count from radon in the air and they are included in the calibration. Even when using short counting intervals, it is not possible for this type of counter to accurately follow radon concentrations that are changing rapidly relative to the 2 to 3 hours time required for the equilibration of ingrowing daughters. However, the counts accumulated in later intervals will tend to compensate for short-term errors in earlier intervals so that this instrument can provide unbiased estimates of averaged concentrations for periods of about a day or more while giving approximations to the more rapidly changing concentrations. A deconvolution routine was reported (Thomas 1972) to establish the correct instantaneous concentrations using shorter counting intervals even when they are rapidly changing. However, the calibrations required for that routine are difficult and that type of routine can still produce significant aberrations in the results for concentrations which are changing significantly within the counting intervals rather than just between intervals (Busigin, Van der Vooren and Phillips 1979).

The Eberline instrument is designed to be used in field applications and is equipped with a weather-resistant cover. It weighs about 110 pounds and is not readily portable. These samplers count radiations at the time the sample is passing through the flask and are sensitive to the alpha particles emitted by ^{220}Rn (thoron) and its short-lived daughters. Because of ^{220}Rn's 56-second half life, it is possible to eliminate ^{220}Rn from the air stream by the use of a charcoal delay bed (not normally supplied with the instrument). Saturation of the charcoal bed should be designed to occur after an interval of about five minutes with subsequent elusion of the radon from the bed. After a period of about 15 minutes of operation, such a delay bed achieves a steady state with its characteristic hold-up time. Since the ^{220}Rn in sampled air is unsupported by its longer-lived parents, this delay would be sufficient to remove about 95% of the thoron from the air stream. It would produce a small delay in the response of the instrument to changing ^{222}Rn concentrations, but for practical purposes the delay is insignificant.

When installing this or other types of continuous samplers, consideration must be given to the moisture and particulate content of the inlet air. There is occasionally a considerable quantity of water exhausted from uranium mine vents when the shaft penetrates an aquifer. Since this water is usually in the form of coarse droplets, it can be removed by using a trap. For extremely large volumes of water, it will be necessary to use a self draining trap with water seals. Because of the limited solubility of radon in water and the relatively small water-to-air ratio, this type of trap should not significantly reduce the radon concentration in the air being sampled. If dust loading is a significant problem, a larger prefilter than is normally supplied with the Eberline unit should be considered.

The RGA 400 radon gas analyzer is a second type of commercially available continuous sequential radon gas analyzer (manufactured by company No. 3 listed in Table 6.1). This analyzer uses electrostatic precipitation to collect the daughters of ^{222}Rn and ^{220}Rn onto the surface of a surface barrier diode detector where their alpha particle emissions can be measured spectroscopically. In this instrument, the radon and thoron progeny consist largely of ions generated in the prefiltered air sample because of radioactive decay. In principal, these ions can be concentrated at the detector. The unit features a programmable microprocessor and is protected from accidental power failure by a battery backup which insures up to ten hours of operation with no data loss in the event of a power failure. This unit also has provision for measurement of the accumulated radon and thoron progeny on the inlet filter so that it can provide a sequential series of measurements of those radionuclides in addition to the radon and thoron. The 1983 price for this unit is approximately $16,900. Since the RGA 400 is a relatively new design, there is very little information available to date concerning its applicability to stack sampling. The electrostatic field collector should be sensitive to the moisture of the sampled air. However, it is our understanding that internal calibrations for the relative humidity are provided in the microprocessor program. The same type of prefiltering and water removal used for the flow-through flask sampler may also be necessary for the RGA sampler. If these preseparators are used, radon progeny measurements will be meaningless, however, and only their radon and thoron measurements can be used.

A third type of continuous sequential sampler that could potentially be modified for stack sampling is a sequential bag sampler. In this design, air is pumped sequentially into an array of bags at a low flow rate. One such unit of the AQS series of samplers of the Aero Vironment Company utilizes Tedlar®[a] bags which are reported by Jackson et al. (1979) to be satisfactory for retaining radon in air samples (manufactured by company No. 1 listed in Table 6.1).

The AQS is not designed as a stack sampler. It is housed in a large barrel with ambient air being collected from a covered ported area at the top of the barrel. The air is then transferred into bags within the barrel using a small pump for each bag. Each pump operated in a programmed sequence. However, Jackson et al. (1979) modified these samplers by attaching an inlet vent

(a) Tedlar® is a trademark of the Dupont Company.

and filtration system between the vent and the instrument. As long as the vent operated, the air velocity was sufficient to rapidly force air through the vent and into the sampled cavity. A sequence of up to twelve bags could be used per 24-hour sampling. Once each day the filled bags were removed and replaced with empty bags and the sequence was repeated. An aliquot of the air from each bag was transferred into an evacuated scintillation flask to permit analysis of the radon content of the sampled air. Thus, this technique could be considered to be sequential or integral sampling coupled with scintillation flask counting. Since Tedlar® bags are rather brittle, it is necessary to be aware of small cracks and leaks that can develop in handling. Bags have a rather limited useful life because they are being regularly evacuated and filled causing wrinkling and ultimately cracking. This type of sequential sampler can be programmed to detect concentration variations in various time intervals depending on the number of bags and switching arrangement that is chosen.

Integrating Radon Samplers

The two principal types of integrating radon samplers that are commercially available are the "Track Etch®"[a] filter cups marketed by Terradex Corporation (manufactured by company No. 7 listed in Table 6.1) and the passive environmental radon monitors (PERMS) marketed by a number of firms. PERMS are designed for environmental use but they are not readily suitable for in-stack applications because of their size and mode of operation. One commercially available PERM is the RDT-310 available from EDA Instruments, Inc. (manufactured by company No. 3 listed in Table 6.1). The unit is a cylinder about 10 in. in diameter and 15 in. in height and contains batteries to permit electrostatic collection of radon progeny at a point near a thermoluminescent chip which is sensitive to alpha particles. These units rely on the passage of air through the housing, then through either a drying agent or a semi-permeable membrane which passes radon but not moisture into a cavity where the radon progeny are collected from the air. These units could possibly be used for stack sampling at some sites if mounted in a separate housing that is ducted to the vent stack. However, this application would significantly increase the cost of the installation.

(a) Track Etch® is a trademark of the Terradex Corporation.

The Track Etch® cup is a simple apparatus which uses (for a radon detector) a film chip that forms detectable tracks by degradation of the chip material along the path of alpha particles. The chip is mounted inside of a plastic cup which has a plastic-coated filter material across its open end. The filter material allows the passage of radon but not its daughters or other particulates into the cup. After several hours, the radon in the cup reaches radioactive equilibrium with its daughters. A portion of the alpha emitting radionuclides in the cup produces tracks in the detector film. After a period of exposure, the cups must be returned to the manufacturer where the detector films are developed by etching to enlarge the tracks. The tracks per unit area are then measured using microscopes and converted to radon concentrations.

These cups have been used extensively for environmental monitoring and exploration work in the uranium industry. The use of the Track Etch® cup in a vent stack would require some precautions in mounting because of the high-velocity air. Special sealing of the filter cap and protection from physical stress would be needed. In addition, since the cup and its cover are constructed of plastic, tests should be performed to assure that the buildup of electrostatic charge from the air stream onto the surfaces does not effect the calibration.

Since units are relatively sensitive, the length of exposure would have to be adjusted to be certain the number of tracks does not exceed the upper limit of readability. Because of the finite time required to reach equilibrium between the internal and external atmospheres, these devices should require at least several days of exposure to conform to the original calibration. It is possible that special calibrations might be necessary for exposures in those vents with extremely high radon concentrations. The use of several grab scintillation flask samples might be more practical for vents emitting extremely high concentrations. For lower concentrations, where standard calibrations can be used, this technique offers the potential of long-term integrated measurements at a number of sampling sites. This can be especially useful when a large-scale program is being considered.

There are no reports addressing the application of Track Etch® to vent stack sampling, and the available reports primarily discuss experiences in home-radon monitoring. Bruno, Israeli and Magno (1983) report that the relative standard deviation of this technique for replicate 1/2 to one-year long measurements in homes in Butte, Montana, is about 15 to 16%. However, they report that when Track Etch®, PERMS, and Eberline continuous sequential radon monitors were compared, the 95% confidence interval of the ratios of simultaneous measurements for any pair of instruments was about +/- 70%. To assure that results are accurate, stack monitoring programs using Track Etch® cups should include at least a few cross checks with another accepted monitoring technique such as multiple grab scintillation flask samples. Replicate analyses should be made at enough vents to determine precision of the technique for that application.

6.1.2 Calibration of Radon Monitoring Instruments

There are presently no nationally- or internationally-recognized standard radon sources. For this reason, major laboratories involved in radon measurements have (for the last two years) been participating in a series of round-robin intercalibration studies. The large-volume test chamber at the Environmental Measurements Laboratory of the Department of Energy has been used to provide a source of air containing moderate levels of ^{222}Rn so that flask grab samplers could be compared. In addition, a special pulse ionization chamber flask has been designed at that laboratory to have theoretically predictable counting efficiency characteristics (a defined, small, dead volume for the flask). On the basis that measurements taken using that flask are accurate, other laboratories have been correcting the efficiencies of their instruments to remove the bias between laboratories (Fisenne, George and Keller 1983). While there are a number of other commercially available radon sources or calibration techniques, calibrations based on them must be considered tentative until a recognized national standard for radon measurement has been established.

At present, two national laboratories have controlled atmosphere test chambers which are being calibrated in relationship to the pulse ionization chamber flask. The Environmental Measurements Laboratory of the U.S. Department of Energy has been selected to provide calibrated test atmospheres which contain ^{222}Rn at environmental levels for calibrating instruments upon request. The U.S. Bureau of Mines, Denver Research Center has been selected to provide the same service with ^{222}Rn concentrations at the occupational levels encountered in uranium mining. These chambers are designed for relatively short-term exposures for a number of users. Long-term exposure calibrations for instruments may need to be done by making a series of short-term measurements at a field site using a calibrated short term type of instrument during an interval that is monitored by the long-term samplers. Then the long-term measurements could be related to the average of the calibrated shorter-term series of measurements.

6.1.3 Vent flow Measurements

To determine the radon emission rate for a vent, the air-flow rate must also be measured. Depending on the face velocity of the air (and other factors), any one of a number of the instruments which are used to monitor high-velocity air in ducts may be used. These include pitot tubes, vane anemometers, hot-wire anemometers and the more costly instruments such as those which make measurements based on the bending of electrical corona discharges, sonar, and the change of electrical capacitance.

In most cases, the field conditions for measurements will be less satisfactory than the ideal conditions which are specified for these instruments, and compromises will be necessary. For example, it will not be possible to use hot-wire anemometers at vents that are emitting a water spray because the readings will be affected by the heat capacity and evaporation of the water. Pitot tubes are not suitable for use unless they can be inserted into the duct. Even then, it is commonly specified that there should be at least ten duct diameters of straight duct upstream and downstream from the measurement point for pitot tube measurements. There are rarely more than five duct diameters of uranium mine vents above the ground. Above-ground sections contain the exhaust fan and flow-straightening fins in close proximity to the measurement point. At some vents, no access holes are available to insert a pitot tube and some type of measurement will have to be taken at the mouth of the vent. One accurate technique would be to measure the time it takes for a tracer injected at the base of a vent to reach the surface if the exact length of the vent hole were known. However, this technique cannot be universally applied because the bases of a number of vents will be in areas bulkheaded to prevent access because of high radon-progeny concentrations and/or the danger of falling rock.

Thus, there are a variety of conditions to cope with when determining the flow rate. The effect of those conditions on each measurement will have to be considered based on past experience or will have to be determined by cross checks between instruments. Some of the more costly techniques may be less susceptible to difficulties from ambient conditions but we have not tested them. Jackson et al. (1980) made some cross checks with vane anemometers and pitot tubes and cross checks between different pitot tubes at different times. Most of the average measurements agreed to within a range of about 20%. There are often severe flow discontinuities in these vent stacks, so that it is typical to make numerous measurements at area increments representative of the cross section and average them. In addition, Jackson et al. (1980) concluded from measurement at two mine vents that temporal variations of the vent flow were relatively small. However, the universal applicability of this limited test was not demonstrated, and it would be best to measure the flow within as short of time as possible, of collecting air samples when making radon measurements.

6.2 EMISSIONS CONTROL DESCRIPTION

The Environmental Protection Agency has considered several options by which their proposed national standard for radionuclide emissions (^{222}Rn) from underground uranium mines could be met.[a] The EPA noted that the development of such an emission standard for underground uranium mines was more difficult and complicated than for other sources emitting radionuclides into air (Federal Register, 1983).

One reason for this difficulty is because there are no ^{222}Rn emissions source control systems presently in use in underground uranium mines upon which to gage a standard, and many proposed control systems were found to be too

(a) The EPA standard limits the annual average ^{222}Rn concentration in air due to emissions from underground mines to 0.2 pCi/ℓ above background in any unrestricted area.

costly and not very effective. The EPA thus concluded that there is no existing practical source control technology for achieving acceptable reductions in ^{222}Rn emissions to air from underground uranium mines.

As alternative control methods, the EPA has suggested the standard be met by any of the following procedures: 1) reducing the percentage of time the mine operates, 2) controlling additional land around the mine site,[a] or 3) increasing the effective height of the release (Federal Register, 1983). The EPA has suggested that perhaps the most effective procedure for limiting ^{222}Rn exposure to individuals might be to provide for a greater dispersion of the released ^{222}Rn from the mine. That is, as per number 3 above, increasing the vent stack height (Federal Register, 1983). Under request by the EPA, the Pacific Northwest Laboratory has estimated the cost of a 20 meter high exhaust borehole vent stack and determined the levelized cost of such vent stacks per ton of ore and per pound of U_3O_8 for a best, worst, and most likely case.

6.2.1 Stacks

Data used in the estimation of the exhaust borehole vent stack cost was obtained from vendors and cost estimating manuals (Means 1982, Engelsman 1981, Page 1977). All necessary equipment was assumed to be rented. Labor rates include: fringe benefits, insurance, taxes, and contractors' overheads and profits. Final dollar values are given in January 1983 dollars. Price data from Engelsman (1981) were escalated up to January, 1982 dollars by increasing the values 6.7% in accord with the Marshall and Swift quarterly equipment cost index for mining and milling (Chemical Engineering 1981, 1982). These price data were then escalated to January 1983 dollars by assuming a 5% increase during 1982.

In estimating the cost it was originally assumed that the stack would be 20 meters high and 4.5 feet in diameter. However, upon further investigation it was found that the most common size exhaust borehole has a diameter of 6.0 feet. Thus, the estimates were scaled up to this size.

(a) Land control cost out to 2 km is discussed in Section 3.8.2, Land Valuation.

Foundation loading was designed to be 850 pounds per square foot. This value is much lower than it normally would need to be. The reason for using such a conservative value is because the soil type and seismic zone are not known. Also, using a low value for foundation loading allows the weight of the stack to be distributed around the cased boreholes. Thus, no engineering or other changes would need to be made to the boreholes.

Wind loading of the stack was also calculated. A value of 20 pounds per square foot was used to determine the load and a safety factor of 67% was added. It was found that the most economical method of support would be the use of guy wires and a quote was obtained for this material.

The major cost component in the estimate is the material of which the stack is made. Prices for two different kinds of stacks were obtained. As mentioned before, the original design has a diameter of 4.5 feet. A quote for this size stack was obtained from Seattle Boiler Works. This design called for the stack to be made of 3/8 inch carbon plate steel. It would be made in two sections (for easy transport) with a flange in the middle and at the bottom. A reinforcing ring two-thirds of the way up was included for attachment of the guy wires. The alternative design was made out of 10 gauge corrugated galvanized spiral steel and prices were obtained from Engelsman (1981).

When scaling up to 6.0 feet it was estimated that the total structure cost would increase in the same proportion as the stack material costs. The cost of the carbon plate steel stack increases linearly with the diameter, thus the price was increased 33%. For the corrugated steel structure it was found that the thickness of the metal would have to be increased to 8 gauge steel and the price would increase 50%. The final cost of the carbon plate steel stack was $16,550 and the cost of the corrugated steel stack was found to be $13,030. A detailed cost estimate appears in Table 6.2.

All information about size, required number of exhaust boreholes, and average mine lifetime production were derived from data for three typical mine types. Specifically, production ranged from 330 to 700 tons per day, and the number of stacks required (of various sizes) was 2 to 13. The years of mine production remaining spanned 2 to 12 years and the average ore grade over the

TABLE 6.2. Exhaust Borehole Stack Cost Final Estimate

Exhaust Borehole Stack Cost Final Estimate	Corrugated Steel Galvanized Spiral	3/8 " Plate Carbon Steel
Excavation		
Backhoe w/operator (1/2 Day)	$ 215.00	$ 215.00
Operating cost @ $5.70 hour	25.00	25.00
Foundation		
6' x 6' concrete reinforced with anchor bolts	320.00	320.00
Stack		
Material	4,430.00	9,020.00 (83$)
Labor to assemble	360.00	
Painting		245.00
Crane w/operator (for assembly and erection)		
55 ton (truck MTD.) hydraulic @ $855/day	1,285.00	855.00
Operating cost @ $16.35/hr	200.00	130.00
Guy Wires		
Cable (coated)	65.00	65.00
Anchor plates, tie rods, and turnbuckles	125.00	125.00
Clamps	80.00	80.00
Erection Labor		
3 struct. steel workers @ $25.50/hr	615.00	615.00
1 struct. steel foreman @ $28.15/hr	225.00	225.00
1 surveyor w/transit for 2 hours	80.00	80.00
Backfill and Compact		
1 Rammer compactor (gas)	45.00	45.00
Operating Cost @ $0.88/hr	10.00	10.00
1 Laborer @ $18.70/hr	190.00	190.00
	$8,270.00	$12,245.00
Convert '82 to '83 dollars (5%)	$8,685.00	$12,410.00
Scale up to 6' (most bore holes are 6' dia.)	~50% (8 vs. 10 gauge) $13,030.00	~33% (still 3/8") $16,550.00

same period varied between 0.15% and 0.40% U_3O_8. A financial analysis was performed to determine the levelized cost of installing the stacks on all exhaust boreholes of a diameter greater than one foot. Development of the financial analysis methodology is included at the end of the report as Appendix G.

Certain costs that are location and/or operation dependent were not included in the basic stack cost estimate. However, these costs are included in the financial analysis and appear below:

- Mobilization and Demobilization of Equipment: This cost would depend on the distance of the mine from available rental equipment, ease of access, and the number of stacks being constructed. Including these costs could easily double the equipment costs and increase the stack cost by 10 to 20 percent.
- Transportation of Labor or Material to and from the Mine: As with the equipment cost, labor costs could be double when including transportation to and from the mine. This would also result in an increased stack cost of 10 to 20 percent. Transportation of materials to the mine could be extremely expensive if it must be fabricated some distance away. For example, the plate steel stack may have to be made hundreds of miles from the mine, which could significantly increase cost.
- Increased Ventilation Cost: This cost would depend for the most part on the length of the borehole. Assuming that the fans could handle the additional load, the only cost would be the increased power consumption. With current prices and assuming 80,000 cfm, increased power consumption of 10% would cost about $4,000 a year per exhaust borehole stack. Under some conditions requiring additional capital investment in the ventilation system, the cost could be considerably greater.
- Prevention of Stack Icing: To prevent icing of the stack during cold weather it would be necessary to either heat the stack or heat the air before it goes into the stack. Assuming it would be necessary to raise the temperature of the exhaust air 10°F and the air

flow was 80,000 cfm, a heater would be required to supply about 24 million Btu per day. Such a heater is available from the Dravo-Hastings Corporation at a list price of $13,157.50. Currently discounts of up to 60% from the list prices are available. At list price the heater is about the same cost as the stack. Assuming the heater was operating about two months a year, the annual operating cost would be approximately $8,640.

The financial analysis included three variations: a best, worst, and most likely case. The assumptions pertaining to each case are listed below:

<u>Best Case</u>

- 4.5 feet diameter corrugated steel stack (30% less than the plate steel stack cost)
- All equipment on site
- All labor on site
- Minimal transportation costs for materials
- No icing problem
- 2.5% increase in ventilation power costs

<u>Worst Case</u>

- 6.0 feet diameter carbon steel stack
- 20% increase in equipment cost for mobilization
- 20% increase in labor cost for transportation to site
- 20% increase in materials cost for transportation to site
- 80% increase in capital cost for heaters
- 10% increase in ventilation power costs
- 4 months per year operation of heaters ($17,280/yr)

<u>Most Likely Case</u>

- 6.0 feet diameter carbon steel stack
- 10% increase in equipment cost for mobilization
- 10% increase in labor cost for transportation to site
- 10% increase in materials cost for transportation to site

- 10% increase in capital cost for heaters
- 5% increase in ventilation power costs
- 2 months per year operations of heaters ($8640/yr)

The results of the financial analysis appear in Table 6.3 as a range of values. The costs are given in dollars per ton of ore as well as dollars per pound of U_3O_8.

TABLE 6.3. Increased Cost of Production Resulting From Adding Ventilation Stacks on Exhaust Boreholes

Best Case	
	\$0.168 - \$0.271 / ton of ore
	\$0.022 - \$0.067 / pound of U_3O_8
Worst Case	
	\$0.724 - \$1.493 / ton of ore
	\$0.091 - \$0.417 / pound of U_3O_8
Most Likely Case	
	\$0.493 - \$0.881 / ton of ore
	\$0.062 - \$0.242 / pound of U_3O_8

6.3 ^{222}Rn EMISSIONS TO AIR FROM UNDERGROUND URANIUM MINES

6.3.1 Methodology of Calculations

The published information that has been located concerning radon emissions from specific underground uranium mines has been limited to two NUREG documents written by Jackson et al. (1979, 1980). In them are reported the results of measurements obtained by grab sampling air emitted from vents at uranium mines. These mines represented 65% of the total U.S. underground U_3O_8 production in 1979. The purpose of this chapter is to review the current status of those mines and to compute an estimate of the current radon emission rate from each of them. Table 6.4, taken from the original Table 6 of the 1980 document summarizes the status of the emissions at that time. The identity of the mines had been coded by agreement with the mine operators at the time permission was granted to make the original measurements. The results will be again reported in a coded fashion since mine operators have requested a continuation of their anonymity.

TABLE 6.4. Summary of Radon Emissions from Underground Mine Vents

Mine	1979 Measurement Ci/yr	1978 Measurement Ci/yr	Overall Average Ci/yr	Ratio 1978-1979
A	7,400 ± 1100		7,400 ± 1100	
B	4,700 ± 60	4,300 ± 100	4,500 ± 300	1.09 ± 0.03
C	5,200 ± 200	3,900 ± 300	4,600 ± 800	1.33 ± 0.11
D	3,630 ± 120		3,630 ± 120	
E	29,800 ± 400		29,800 ± 400	
F	9,200 ± 270	9,500 ± 200	9,400 ± 200	0.97 ± 0.03
G	2,150 ± 50	1,460**	1,800 ± 400	1.47 ± 0.03
H	15,200 ± 300		15,200 ± 300	
I	1,690 ± 80		1,690 ± 80	
J	7,760 ± 190	8,100 ± 400	7,900 ± 200	0.96 ± 0.05
K	7,000 ± 190	5,870**	6,400 ± 700	1.19 ± 0.03
L	1,470 ± 40	1,320 ± 30	1,400 ± 300	1.11 ± 0.05
M-Q	Not Sampled			
R	15,000 ± 400	14,600	14,800 ± 300	1.03 ± 0.04
S	Not Sampled			
T	1,890 ± 120		1,890 ± 120	
U	890 ± 20		890 ± 20	
V	1,010 ± 60		1,010 ± 60	
W,X	Not Sampled			
Y	17,500 ± 400		17,500 ± 400	
Z		2,640 ± 70	2,640 ± 70	
AA	2,100**	1,490 ± 70	1,800 ± 400	1.41
BB	2,130 ± 80	1,840 ± 70	2,000 ± 200	1.16 ± 0.06
CC		2,120 ± 50	2,120 ± 50	
DD		960 ± 40	960 ± 40	
EE	6,500 ± 70		6,500 ± 70	
FF	2,510 ± 80		2,510 ± 80	
GG	190 ± 7	146 ± 3	170 ± 30	1.30 ± 0.05
HH	1,040 ± 60		1,040 ± 60	
II	470 ± 10		470 ± 10	
		SUM ALL MINES ± STD. DEV.	15,000 ± 2000 (± 3000)	1.18 ± 0.05 AVE

* Single sample
** Average of sequential sample data, 1978

Source: Jackson, et al. (1980)

To update the status, a representative of each mine operator was contacted by telephone (on or before August 16, 1983) and asked if their mine(s) were still operating. If a complete or partial shutdown was indicated, then the operator was asked if the mine vents and portals were operating or sealed. If a mine was shut down and an attempt made to seal all vents, ^{222}Rn emissions to the surface were considered to be zero. If a mine was shutdown but with vents still operating either continuously or intermittently (i.e., not sealed), the mine was considered to be emitting radon.

For the shutdown but radon emitting mines, no attempt was made to quantify the reduction in radon emissions resulting from the decay of radionuclides in stagnated underground air. The reduction of radon due to decay is dependent on elapsed time of air stagnation which in turn depends on cycles of fan operation and the rate of natural draft ventilation.

For operating mines, the current emission rate will be considered as greater than the rate measured during the 1979 study because of a greater mine wall surface area exposed by continuing mining. The rate of increase will be estimated for those mines that are operating in 1983 and that provided annual production rate data in the 1979 study (see Table 1 of Jackson, et al. 1980). In addition, new data have been received from these current operating mines.

Some mine operators indicated that their current mine emissions should be reduced from those measured in 1978 and 1979 because the current mine production rate is smaller and/or that areas of their mine are inactive or not being ventilated. However, except where specific details about the bulkheading and other measures taken to prevent the passage of radon to the surface were available, it was not possible to predict whether radon from the unused areas of a mine is decaying underground or if it is reaching the surface. Since that information has not been received for these mines in question at the time of this report, radon emissions from these mines will be calculated using the same assumptions as for operating mines.

Table 6.5 shows the current status of the mines from the 1978-1979 study. Based on the Mine Survey data in Table 1 of the 1980 report, the total daily ore production in 1979 of the mines which are still operating today was 8500 tons. This is equivalent to 2,125,000 tons ore per year (based on 250

TABLE 6.5. Current (1983) Status of Mines Monitored in 1978-1979 Survey of Jackson, et al. (1979, 1980)

Mine	Status: Operating	Status: Shutdown with Vents Sealed
A	X	
B	X	
C	X	
D	X	
E	X	
F	X	
G	X	
H		X
I	X	
J		X
K	X	
L		X
M-Q	Not Sampled	
R		X
S	Not Sampled	
T	X	
U		X
V		X
W-X	Not Sampled	
Y		X
Z		X
AA		X
BB		X
CC		X
DD		X
EE		X
FF		X
GG		X
HH		X
II		X

operating days per year) and is 59% of the total production of 3,600,000 tons ore per year of all mines where vents were monitored in 1979 and 1980.

It is difficult to forecast the future status of these mines, given the economic circumstances of the uranium industry. For the purposes of predicting future emissions, it will be assumed that the mines in the survey, operating in 1983, will remain open through 1990, and that economics will dictate production rate changes in proportion to the changes forecast for the entire industry by the Department of Energy (1983).

The prediction of current and future ^{222}Rn radon emission rates from measurements taken 4 to 5 years ago, requires simplifying assumptions that are not necessarily accurate for individual mines. Those assumptions have been discussed in the section of this report covering techniques for monitoring radon emissions. Basically, it is assumed that each mine will increase in size by an increment of annual ore production and that the radon emission rate will also increase. Using the relationship between the cumulative production of ore and the radon emission rate (0.44×10^{-2} Ci/year ton of ore) determined by Jackson et al. (1980), one can predict the annual increase of radon emissions from the annual production listed for each mine in Table 1 taken from the same document. This emission rate coefficient was computed for the total emissions from both above-ground sources and ventilation areas. However, since ventilation air was estimated to be 98% of the total emanation rate reported for 1978 and 1979, the error caused by above-ground sources is negligible.

It is unlikely that the annual production rate from each mine remained fixed during its life in the years following 1978. Since new production statistics for each mine have not currently been received, the 1978 production rates are adjusted in the current calculation to reflect the ratio of the national total production statistics for all underground mines in a given year to that for 1978 as given by the Department of Energy (DOE 1983). These data are summarized in Table 6.6. Also shown in this table, in column four, are the products of the predicted fractions of 1978 production for each year multiplied by the emission rate coefficient for 1978-1979. Multiplying the 1978 annual production rate for a mine by the factors in column four will yield an estimate for the annual increments of radon emission rate to be added to the measured

TABLE 6.6. Uranium Ore Produced from Underground Mines and Fractional Radon Emission Rate Factors

Year	Ore Production (thousands tons)	Fraction of 1978 Production	Fractional Radon Emission Rate Factor (Fraction of 1978 x Ci/yr ton)
1978	6,105	1.00	0.44×10^{-2}
1979	5,356	0.88	0.39×10^{-2}
1980	6,351	1.04	0.46×10^{-2}
1981	5,229	0.86	0.38×10^{-2}
1982	2,809	0.46	0.20×10^{-2}
1983(a)	2,208	0.36	0.16×10^{-2}
1984(a)	2,011	0.33	0.15×10^{-2}
1985(a)	1,719	0.28	0.12×10^{-2}
1986(a)	1,695	0.28	0.12×10^{-2}
1987(a)	1,672	0.27	0.12×10^{-2}
1988(a)	1,901	0.31	0.14×10^{-2}
1989(a)	1,874	0.31	0.14×10^{-2}
1990(a)	1,846	0.30	0.13×10^{-2}
Σ 1979 through 1983	21,953	3.60	1.58×10^{-2}
Σ 1979 through 1990	34,671	5.68	2.50×10^{-2}

(a) Uranium ore produced from underground mines is based on an estimated total industry U_3O_8 production for 1983 of 10,600 tons and for 1990 of 9600 tons. For intermediate years annual U_3O_8 production rates were interpolated between the 1983 and 1990 values. The conversion from total industry tons U_3O_8 to total industry tons ore was made using average ore grades predicted for each year obtained via personal communication from the DOE Grand Junction office. Finally, the percentage of total industry ore attributed to underground mining operations was estimated at 25% for the years between 1983-1990. This percentage was estimated from the trend in historical data from the DOE (1983) on underground versus open-pit mining.

radon emission rate in 1978 for that mine. The sum of these factors for years 1979 through 1983 or 1990 yields the expected added increment of ^{222}Rn emission during the intervals between the original measurement set of Jackson et al. (1980) and December 31, 1983 or December 31, 1990, respectively.

6.3.2 Calculation Process

The calculation process for predicting 1983 and 1990 ^{222}Rn emission rates for each mine in the 1978-79 Jackson et al (1980) study will be outlined.

First, the mine's given 1978 daily ore rate is converted to an annual rate:

$$(DPR)_i \ (OPD) = (APR)_i \qquad (6.1)$$

where

DPR = 1978 daily production rate per mine $_i$ in tons ore/day

OPD = 250 operating days per year

APR = 1978 annual production rate per mine $_i$ in tons ore/day

Then by using the calculated cumulative fractional radon emission rate factor for 1979-1983 and 1979-1990 (see Table 6.6), a cumulative increase of annual rates of ^{222}Rn emissions per mine since 1978 can be obtained:

$$(APR)_i \ (CFRE)_x = (CARE)_i \qquad (6.2)$$

where

CFRE = cumulative fractional radon emission rate factor for period x (1979-1983 and 1979-1990)

CARE = cumulative increase of annual ^{222}Rn emission rates per mine i since 1978, in Ci/yr.

Finally, the $(CARE)_i$ figure is added to the mine's 1978 reported ^{222}Rn emission rate:

$$(CARE)_i + (RER)_i = (TRER)_i \qquad (6.3)$$

where

RER = 1978 ^{222}Rn emission rate per mine i in Ci/yr.

TRER = new total ^{222}Rn emission rate per mine i as of December 31, 1983 and December 31, 1990 in Ci/yr.

The result $(TRER)_i$ is the predicted total ^{222}Rn emission rate for each mine at the end of years 1983 and 1990.

6.3.3 Results and Discussion

The results of this computation process for all the mines sampled in 1978 and 1979 are shown in Table 6.7. These computed extrapolations predict that the total emission rate for the mines sampled will be 70% of the 1978 and 1979 rate on December 31, 1983 and 83% of the 1978 and 1979 rate on December 31, 1990. This assumes that all the mines for which computations were made continue to operate, and that their production follows the industry averages. It can be argued that the production cutbacks observed in the national totals occur as a result of mine closures. However, in discussions with representatives of operating mine companies, a frequently heard comment was that production from the mines was being significantly reduced at the present time. Moreover, several new mines have started producing since the 1978 and 1979 surveys. Whether these new mines represent a significant fraction of the production lost from closed mines is not known at the present time. The radon produced by a new mine would initially be expected to be lower than a similarly sized operation that was closing after several years of productive life. It can be seen in Table 6.7 that the predicted emission rate increases much more rapidly for the four-year-old mine G than for a 20-year-old mine such as F.

6.3.4 Conclusions

The model used for predicting future emissions from past measurements uses only the trend observed in the data of the 1978 and 1979 study, which predicts an ever increasing release of ^{222}Rn from a mine unless areas are completely sealed from the surface. Since actual current production statistics were not available for each mine, and the suitability of the model to individual operations has not been tested, these predictions must be viewed as tentative and suitable only for industry-wide predictions. Based on the predictions, radon emission rates for individual operating mines are increasing but a significant fraction of mines is no longer operating and most of these mines are sealed. Additional onsite inspection of the degree of sealing for "closed mines" should be performed to verify that bulkheads and seals are adequate. Current measurements at operating mines, along with production statistics up to the present time, should be collected to demonstrate the suitability of the model to industry-wide projections. However, only a detailed research program could improve

TABLE 6.7. Measured and Predicted ^{222}Rn Emission Rates

Mine	Production Rate (hundred thousand tons/yr)	Emission Rate on 12-31-78 (thousands Ci/yr)	Predicted Emission Rate on 12-31-83 (thousands Ci/yr)	Predicted Emission Rate on 12-31-90 (thousands Ci/yr)
A	5.5	7.4	16.0	21.1
B	1.78	4.5	7.3	9.0
C	2.36	4.6	8.3	10.5
D	2.68	3.63	7.9	10.3
E	2.50	29.8	33.8	36.1
F	1.79	9.4	12.2	13.9
G	1.98	1.80	4.9	6.8
H	1.20	15.2	0	0
I	0.75	1.69	2.88	3.57
J	0.92	7.9	0	0
K	0.88	6.4	7.8	8.6
L	0.62	1.4	0	0
M-Q		Not Sampled		
R	0.28	14.8	0	0
S		Not Sampled		
T	1.05	1.89	3.55	4.2
U	1.25	0.89	0	0
V	1.38	1.01	0	0
W-X		Not Sampled		
Y	6.6	17.5	0	0
Z	1.25	2.64	0	0
AA	--	1.80	0	0
BB	--	2.00	0	0
CC	--	2.12	0	0
DD	--	0.96	0	0
EE	--	6.50	0	0
FF	--	2.51	0	0
GG	--	0.17	0	0
HH	--	1.04	0	0
II	--	0.47	0	0
Total		150.	105.	124.

the model by adding the significant parameters in addition to the quantity of ore extracted. Again, we caution that the predicted December, 1983 emission rates are subject to large uncertainties and that actual field measurements are required to obtain accurate emission rates.

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APPENDIX A

GLOSSARY OF MINING TERMS

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GLOSSARY OF MINING TERMS

back	The roof or upper part of any underground mining cavity.
cribbing	The construction of cribs, or timbers laid at right angles to each other, sometimes filled with earth, as a roof support or as a support for machinery. The close setting of timber supports when shaft sinking through loose ground. The timber is usually square or rectangular and practically no ground is exposed. The method is also used for constructing ore chutes.
cross-cut	A horizontal opening driven across the course of vein or, in general, normal to the direction of main workings.
drift	A horizontal opening in or near an ore body and parallel to the course of the vein or the long dimension of the ore body.
muck	Rock or ore broken in the process of mining.
raise	A vertical or inclined opening driven upward from a level to connect with the level above, or to explore the ground for a limited distance above one level.
roof	The ceiling of any underground excavation. Same as the "back."
roof bolt	Long steel bolts driven into walls or roof of underground excavations to strengthen the pinning of rock strata. They are expanded by means of a wedge which opens a sleeve surrounding it.
room and pillar method	In coal and metal mining, supporting the roof by pillars left at regular or irregular intervals.
sand fill	Hydraulic or pneumatic filling, stowing. Use of sand conveyed underground by water or air to support cavities left by extraction of ore.
set	A timber or steel frame for supporting the sides of an excavation, shaft or tunnel.
adit	Nearly horizontal passage from the surface.
shaft collar	Supporting framework at top of shaft from which linings may be hung. The term applies to the timber, steel, or concrete around the mount or top of a shaft.
sill	The floor of an opening or passage in a mine.

Glossary of Mining Terms (contd)

<u>slusher</u> — A machine used for loading coal or rock by pulling an open-bottomed scoop back and forth between the face and the loading point by means of ropes, sheaves, and a multiple drum hoist.

<u>square set stoping</u> — A method of stoping in which the walls and back of the excavation are supported by regular framed timbers forming a skelton enclosing a series of connected, hollow, rectangular prisms in the space formerly occupied by the excavated ore and providing continuous lines of support in three directions at right angles to each other. The ore is excavated in small, rectangular blocks just large enough to provide room for standing a set of timber.

<u>stope</u> — Commonly applied to the extraction of ore, but does not include the ore removed in sinking shafts and in driving levels, drifts, and other development openings.

Source: New Mexico Energy and Minerals Department.

APPENDIX B

PRODUCTIVITY FACTOR CALCULATION

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PRODUCTIVITY FACTOR CALCULATION

The productivity factor is calculated as follows:

$$\frac{4{,}274{,}928 \text{ total 4th qtr., 1982 employee-hours 29 study mines less 4 study mines known either not yet producing or shut down prior to 1982}}{0.84 \text{ fraction of production represented by 29 study mines}} = 5{,}089{,}200 \text{ 1982 total hours, underground uranium mining industry}$$

$$\frac{5{,}089{,}200 \text{ hrs}}{2000 \text{ hrs equalling one FTE worker}} = 2545 \text{ 1982 total FTE workers, underground uranium mining industry}^{(a)}$$

$$\frac{6200 \text{ 1982 total tons } U_3O_8 \text{ for underground mining industry}}{2545 \text{ FTE workers}} = 2.44 \text{ tons } U_3O_8 \text{ per FTE worker per year, underground uranium mining industry.}$$

(a) The actual total numbers of underground uranium "miners" (1982 average employment level) as reported by DOE (1983) is 1275. This much lower, average number is probably due to the "substantial layoffs which occurred late in 1982" as noted by the DOE. Additionally, the DOE breaks down the total underground employment (2150) into "miners" (1275) and "service and support" (875). The MSHA (1982) data does not make such a breakdown distinction in its employee-hours, "workers," listing.

APPENDIX C

STUDY MINE DATA SHEETS

APPENDIX C

MINE NAME: Hacks Canyon #1 and #2

COMPANY: Energy Fuels Nuclear, Inc.

LOCATION: T37N, R5W, N1/2 Sec. 26, Mohave County, Arizona

OPERATING STATUS: Active as of 2/83, although Energy Fuels Nuclear suspended milling operations on 1/21/83 (E & MJ 1983).

DESIGNATED MINE CAPACITY: 300-500 tons ore per day[a]

ANNUAL PRODUCTION: N.A.[b]

CUMULATIVE PRODUCTION: 1950, '52, '53, '54, '64 - Hacks #1 (?) produced 1,329 tons ore at 0.18% U_3O_8 (Scarborough 1981).

RESERVES: There is a possible 500,000 tons of uranium-copper ore or about 3 million pounds U_3O_8 (new discovery as announced by Energy Fuels Nuclear, December, 1980, for Hacks Canyon #2) (Scarborough 1981).

DEPTH: N.A.

GRADE:
Hacks Canyon #1:
- 0.009 - 1.798% U_3O_8 (Scarborough 1981)
- 0.50% U_3O_8[a]

Hacks Canyon #2:
- 0.30% U_3O_8 (Scarborough 1981)
- could be as high as 1.0% U_3O_8[a]

AGE OF MINE: first ore shipment in 1981 from Hacks #2.[a]

OTHER COMMENTS:

- Energy Fuels Nuclear has a mill in Blanding, Utah, which opened in 1980. It is a 2000 st/d operation. Mill feed is from the two Hacks Canyon Mines and other toll milling sources (E & MJ 1983)
- R. B. Scarborough says that Energy Fuels Nuclear is a privately owned company by a group of individuals. Its president, Bob Adams, died recently and the Company has since slowed down somewhat. He says that Energy Fuels Nuclear is a very secretive company as to their exploration techniques and deposit statistics. Energy Fuels has the strip of county north of the Grand Canyon fairly tied up.[a]

(a) Information from R. B. Scarborough, Arizona Bureau of Geology and Mineral Technology, Tucson, Arizona.
(b) N.A. = not available

MINE NAME: Pigeon Mine

COMPANY: Energy Fuels Nuclear, Inc.

LOCATION: T38N, R2W, N1/2 Sec. 5, Coconino County, Arizona

OPERATING STATUS: As of 2/83 the mine is still under construction. Production and ventilation shafts are completed, the mine is being readied for future production (E & MJ 1983).

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: none

CUMULATIVE PRODUCTION: none

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: not yet producing[a]

OTHER COMMENTS: The Pigeon Mine has a 1400 foot vertical shaft. A second shaft is now being drilled.[a]

(a) Information from Bill Chenoweth, U.S. Department of Energy, Grand Junction, Colorado.

MINE NAME: Kanab North

COMPANY: Energy Fuels Nuclear, Inc.

LOCATION: T38N, R3W, SW1/4 Sec. 17, Mohave County, Arizona

OPERATING STATUS:
- Construction bids have been solicited, but no construction has been done to date.[a]
- The company recently got permission from the BLM to begin sinking a shaft.[b]

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: none

CUMULATIVE PRODUCTION: none

RESERVES: Unknown by Energy Fuels Nuclear. They are sinking the shaft from limited drill data and assumptions that the ore body exists from their geologic model.[b]

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: not yet producing[b]

OTHER COMMENTS:

(a) Information from W. I. Enderlin, Battelle Pacific Northwest Laboratories, Richland, Washington.
(b) Information from Bill Chenoweth, U.S. Department of Energy, Junction, Colorado.

MINE NAME: Sunday

COMPANY: Union Carbide Corporation

LOCATION: T44N, R18W, SE1/4 Sec. 13, San Miguel County, Colorado

OPERATING STATUS: active as of 11/82 (U.S. Dept. of Labor 1982)

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1980: underground mines feeding Uravan mill produced 100,000-500,000 tons ore (E & MJ 1981).

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: King Solomon

COMPANY: Union Carbide Corporation

LOCATION: T48N, R17W, NW1/4 Sec. 19, Montrose County, Colorado

OPERATING STATUS: active as of 11/82 (U.S. Dept. of Labor 1982).

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1980: underground mines feeding Uravan mill produced 100,000-500,000 tons ore (E & MJ 1981).

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Deremo-Snyder

COMPANY: Union Carbide Corporation

LOCATION: T42N, R20W, S 1/2 Sec. 2, San Miguel County, Colorado

OPERATING STATUS: Active as of 11/82 (U.S. Dept. of Labor 1982).

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1980: underground mines feeding Uravan mill produced 100,000-500,000 tons ore (E & MJ 1981).

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Mt. Taylor

COMPANY: Gulf Mineral Resources

LOCATION: T13N, R8W, SE 1/4 Section 24, Cibola County, New Mexico

OPERATING STATUS: Placed on standby status in November 1982, with support facilities being maintained (Gulf Oil Corp. 1982).

DESIGNED MINE CAPACITY: Expected production of 4,000-4,500 tons/day or 1.42 million tons ore/year (Perkins 1979).

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: N.A.

RESERVES: 125 million lb U_3O_8[a]

DEPTH:
- 3,100 ft (Energy and Minerals Dept 1982)
- ~3,300 ft (Perkins 1979)

GRADE: Average grade of 0.25-0.30% U_3O_8[a]

AGE OF MINE: Mining scheduled to begin 1979, with 20 year expected life (Perkins 1979).

OTHER COMMENTS:
- Were producing until late 1982; toll milling at Kerr-McGee mill.[a]
- There is reported significant quantities of Molybdenum and Vanadium in the ore. Rock face temperatures are expected to reach 130°F, requiring air conditioning and worker's to possibly wear ice vests (Perkins 1979).
- Gulf hasn't sold any U_3O_8 to date from Mt. Taylor. The company has a $419 million investment in the Mt. Taylor mine complex as of 12/31/82. Development will resume when uranium demand improves and market prices return to economic levels (Gulf Oil Corp. 1982).

(a) Information from C. Wentz, Uranium Specialist, Energy and Minerals Department, Santa Fe, New Mexico.

MINE NAME: Old Church Rock

COMPANY: UNC, Resources, Inc.

LOCATION: T16N, R16W, N 1/2 Section 17, McKinley County, New Mexico

OPERATING STATUS: Placed on standby in April, 1982. Decision made to allow the mine to flood (UNC Resources, Inc. 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 - 0.6 million lb U_3O_8 produced in concentrate from UNC's Church Rock Mines (Old Church Rock and Church Rock NE) (UNC Resources, Inc. 1982)

CUMULATIVE PRODUCTION:

- 77,965 tons ore yielding 302,608 lb U_3O_8 (0.19% U_3O_8) until 1970.[a]
- U.S. Department of Energy production class of 200,000 to 2 million lb U_3O_8.[a]

RESERVES: 33.6 million tons ore material containing 57.3 million lb U_3O_8 recoverable by conventional mining methods (as of 12/31/82). Reserve figure includes the following mines:

2 Church Rock Mines (on standby)
3 Ambrosia Lake Mines (all on standby status)
St. Anthony Mines, Laguna area (on standby) (UNC Resources, Inc. 1982)

DEPTH: Shaft #1 - 862 ft, Shaft #2 - 800-900 ft[a]

GRADE: Figured to be about 0.10% (calculated from reserves data in annual report) (UNC Resources, Inc. 1982)

AGE OF MINE: Years of operation: 1960-62, 1976-77, and 1979-82[a]

OTHER COMMENTS: Previously mined in the Dakota and Westwater stratigraphic units of the Morrison Formation from a vertical shaft by Phillips Petroleum Company in 1960-1962 (Perkins 1979).

(a) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Church Rock Northeast

COMPANY: UNC Resources, Inc.

LOCATION: T17N, R16W, Section 35, McKinley County, New Mexico

OPERATING STATUS: Placed on standby in April, 1982. Decision made to allow the mine to flood (UNC Resources, Inc. 1982)

DESIGNED MINE CAPACITY: Up to 3,000 tons/day (Perkins 1979)

ANNUAL PRODUCTION: 1982 - 0.6 million lb. U_3O_8 produced in concentrate from UNC's Church Rock Mines (Old Church Rock and Church Rock NE) (UNC Resources, Inc. 1982)

CUMULATIVE PRODUCTION: N.A.; U.S. Department of Energy production class of 2-20 million lb U_3O_8.[a]

RESERVES:
- 33.6 million tons ore material containing 57.3 million lb U_3O_8 recoverable by conventional mining methods (as of 12/31/82). Reserve figure includes the following mines:
  2 Church Rock Mines (on standby)
  3 Ambrosia Lake Mines (all on standby status)
  St. Anthony Mines, Laguna area (on standby) (UNC Resources, Inc. 1982)
- 20 million tons ore (60 million lb U_3O_8) (Perkins 1979)
- 15 million lb U_3O_8 at 0.15% U_3O_8 estimated in 1969.[a]

DEPTH:
- 1,700 ft (Energy and Minerals Dept. 1982)
- 1,793 ft[a]

GRADE: Average 0.15% U_3O_8 (Perkins 1979).

AGE OF MINE: Began production in 1972[a]

OTHER COMMENTS: The mine has two vertical shafts and five exhaust fans with an air discharge of 667,500 (ACFM) (Perkins 1979).

(a) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Church Rock #1

COMPANY: Kerr-McGee Corporation

LOCATION: T17N, R16W, Section 35, McKinley County, New Mexico

OPERATING STATUS: Producing as of 7/16/82 (Energy and Minerals Dept. 1982)

DESIGNED MINE CAPACITY: Production goal of 1,000 tons per day (Perkins 1979)

ANNUAL PRODUCTION: 1982: 1,170,473 tons of company mined ore were processed at the Ambrasia Lake mill at an average grade of 0.174% (ore from the Ambrosia Lake and Church Rock Mines) (Kerr-McGee Corp. 1982).

CUMULATIVE PRODUCTION: N.A.; U.S. Department of Energy production class of 2-20 million lb U_3O_8.[a]

RESERVES: At 6 foot/0.10% cut off, Church Rock ore body (combined Church Rock #1 and Church Rock East Mines) - 2,345,000 tons ore containing 10,168,000 lb U_3O_8. These reserves "relate only to those estimated quantities of uranium - bearing material which are available to mines actively producing ... and which are believed capable of being profitably mined and sold under present technology, regulatory and economic conditions" (Kerr-McGee Corp. 1982).

DEPTH:
- Maximum depth of 1,851 ft (Perkins 1979)
- 1,673-1,755 ft deep (Energy and Minerals Dept. 1982)

GRADE: 0.22% U_3O_8 (Kerr-McGee Corp. 1982)

AGE OF MINE:
- Began production 1973 or 1974,[b]
- Another communication says 1976[a]

OTHER COMMENTS:
- The mine has one vertical shaft and four exhaust fans with an air discharge of 392,000 (ACFM) (Perkins 1979).
- In 1978, the mine waste bench was over 30 acres. Mined rock of less than 0.03-0.05% U_3O_8 goes to the waste bench. Heat output of the mine is about 350,000 BTU's per minute. The initial development cost has been $15 million (Perkins 1979).

(a) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.
(b) Information from C. Wentz, Uranium Specialist, Energy and Minerals Department, Santa Fe, New Mexico.

MINE NAME: Church Rock East

COMPANY: Kerr-McGee Corporation

LOCATION: T17N, R16W, Section 36, McKinley County, New Mexico

OPERATING STATUS: Producing as of 7/16/82 (Energy and Minerals Dept. 1982)

DESIGNED MINE CAPACITY: Originally 500 tons/day estimated; 1979 - production increased to ~1500 tons/day (Perkins 1979).

ANNUAL PRODUCTION: 1982: 1,170,473 tons of company mined ore were processed at the Ambrosia Lake mill at an average grade of 0.174% (ore from the Ambrosia Lake and Church Rock Mines) (Kerr-McGee Corp. 1982)

CUMULATIVE PRODUCTION: N.A.; U.S. Department of Energy production class of 200,000 - 2 million lb U_3O_8.[a]

RESERVES: At 6 foot/0.10% cut off, Church Rock ore body (combined Church Rock #1 and Church Rock East Mines) - 2,345,000 tons ore containing 10,168,000 lb U_3O_8. These reserves "relate only to those estimated quantities of uranium - bearing material which are available to mines actively producing ... and which are believed capable of being profitably mined and sold under present technology, regulatory and economic conditions" (Kerr-McGee Corp. 1982).

DEPTH:
- 1,545 ft expected depth (Perkins 1979)
- 1,635 ft[a]

GRADE: 0.22% U_3O_8 (Kerr-McGee Corp. 1982)

AGE OF MINE: Began production in 1979[b]

OTHER COMMENTS: In 1979, the east vent hole was enlarged to be used for hoisting, decreasing the haulage distance for ore mined from the east ore body (Perkins 1979).

(a) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.
(b) Information from C. Wentz, Uranium Specialist, Energy and Minerals Department, Santa Fe, New Mexico.

MINE NAME: Kerr-McGee Section 30E

COMPANY: Kerr-McGee Corporation

LOCATION: T14N, R9W, E 1/2, Section 30, McKinley County, New Mexico

OPERATING STATUS: Active as of 2/83, but at reduced production levels.[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 reported company mined ore processed at Ambrosia Lake mill was 1,170,473 tons (includes all Ambrosia Lake Mines and two Church Rock Mines) (Kerr-McGee Corp. 1982).

CUMULATIVE PRODUCTION:

- The Ambrosia Lake mill, in 24 years, has processed 33 million tons of ore containing 125 million lb U_3O_8 (Kerr-McGee Corp. 1982).
- U.S. Department of Energy production class of 2-20 million lb U_3O_8;[b]
- 2,855,164 tons ore yielding 15,064,956 lb U_3O_8 (0.26%) until 1970.[b]

RESERVES: At 5 operating mines in 1982 (Section 19, 30, 30W, 35, 36) 3,925,000 tons ore containing 17,105,000 lb U_3O_8 (Kerr-McGee Corp. 1982).

DEPTH:

- 656 ft (Energy and Minerals Dept. 1982)
- 750 ft[b]

GRADE: 0.22% U_3O_8 (average for five mines this study) (Kerr-McGee Corp. 1982)

AGE OF MINE: Began production in 1959[b]

OTHER COMMENTS:

- The mine has one vertical shaft, and 11 exhaust fans with air discharge of 348,500 (ACFM) (Perkins 1979).
- At the Kerr-McGee's Ambrosia Lake properties, approximately 12% of the U_3O_8 is held by mining claims on Federal lands which are owned or leased, while the remainder of the U_3O_8 is held under mining leases on fee lands (Kerr-McGee Corp. 1982).

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.

(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Kerr-McGee Section 30W

COMPANY: Kerr-McGee Corporation

LOCATION: T14N, R9W, W 1/2, Section 30, McKinley County, New Mexico

OPERATING STATUS: Active as of 2/83, but at reduced production levels[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 reported company mined ore processed at Ambrosia Lake mill was 1,170,473 tons (includes all Ambrosia Lake Mines and two Church Rock Mines) (Kerr-McGee Corp. 1982).

CUMULATIVE PRODUCTION:
- The Ambrosia Lake mill, in 24 years, has processed 33 million tons of ore containing 125 million lb U_3O_8 (Kerr-McGee Corp. 1982).
- U.S. Department of Energy production class of 2-20 million lb U_3O_8;[b]
- 68,895 tons ore yielding 282,714 lb U_3O_8 (0.21%) until 1970.[b]

RESERVES: At 5 operating mines in 1982 (Section 19, 30, 30W, 35, 36) 3,925,000 tons ore containing 17,105,000 lb U_3O_8 (Kerr-McGee Corp. 1982).

DEPTH:
- 701-740 ft (Energy and Minerals Dept. 1982)
- 802 ft[b]

GRADE: 0.22% U_3O_8 (average for five mines this study) (Kerr-McGee Corp. 1982)

AGE OF MINE: Began production in 1969[b]

OTHER COMMENTS:
- Vertical shaft, four exhaust fans with air discharge of 265,000 (ACFM) (Perkins 1979).
- At the Kerr-McGee's Ambrosia Lake properties, approximately 12% of the U_3O_8 is held by mining claims on Federal lands which are owned or leased, while the remainder of the U_3O_8 is held under mining leases on fee lands (Kerr-McGee Corp. 1982).

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.
(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Kerr-McGee Section 19

COMPANY: Kerr-McGee Corporation

LOCATION: T14N, R9W, Section 19, McKinley County, New Mexico

OPERATING STATUS: Producing as of 7/16/82 (Energy and Minerals Dept. 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 reported company mined ore processed at Ambrosia Lake mill was 1,170,473 tons (includes all Ambrosia Lake Mines and two Church Rock Mines (Kerr-McGee Corp. 1982)

CUMULATIVE PRODUCTION:

- The Ambrosia Lake mill, in 24 years, has processed 33 million tons of ore containing 125 million lb U_3O_8 (Kerr-McGee Corp. 1982).
- U.S. Department of Energy production class of 2-20 million lb U_3O_8.[a]

RESERVES: At 5 operating mines in 1982 (Section 19, 30, 30W, 35, 36) 3,925,000 tons ore containing 17,105,000 lb U_3O_8 (Kerr-McGee Corp. 1982)

DEPTH:

- 640-705 ft (Energy and Minerals Dept. 1982)
- 784 ft[a]

GRADE: 0.22% U_3O_8 (average for five mines this study) (Kerr-McGee Corp. 1982)

AGE OF MINE: Began production 1976[a]

OTHER COMMENTS:

- The mine has six exhaust fans, with air discharge of 205,000 (ACFM) (Perkins 1979).
- At the Kerr-McGee's Ambrosia Lake properties, approximately 12% of the U_3O_8 is held by mining claims on Federal lands which are owned or leased, while the remainder of the U_3O_8 is held under mining leases on fee lands (Kerr-McGee Corp. 1982).

(a) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Kerr-McGee Section 35

COMPANY: Kerr-McGee Corporation

LOCATION: T14N, R9W, Section 35, McKinley County, New Mexico

OPERATING STATUS: Active as of 2/83, but at reduced production levels[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 reported company mined ore processed at Ambrosia Lake mill was 1,170,473 tons (includes all Ambrosia Lake Mines and two Church Rock Mines) (Kerr-McGee Corp. 1982).

CUMULATIVE PRODUCTION:
- The Ambrosia Lake mill, in 24 years, has processed 33 million tons of ore containing 125 million lb U_3O_8 (Kerr-McGee Corp. 1982).
- U.S. Department of Energy production class of 2-20 million lb U_3O_8.[b]

RESERVES: At 5 operating mines in 1982 (Section 19, 30, 30W, 35, 36) 3,925,000 tons ore containing 17,105,000 lb U_3O_8 (Kerr-McGee Corp. 1982).

DEPTH:
- 1,186-1,336 ft (Energy and Minerals Dept. 1982)
- 1,398 ft[b]

GRADE: 0.22% U_3O_8 (average for five mines this study) (Kerr-McGee Corp. 1982).

AGE OF MINE: Began production 1971[b]

OTHER COMMENTS:
- The mine has one vertical shaft, and six exhaust fans with air discharge of 414,000 (ACFM) (Perkins 1979).
- At the Kerr-McGee's Ambrosia Lake properties, approximately 12% of the U_3O_8 is held by mining claims on Federal lands which are owned or leased, while the remainder of the U_3O_8 is held under mining leases on fee lands (Kerr-McGee Corp 1982). This mine was originally called the Elizabeth Mine.

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Mi[illegible]n, New Mexico.
(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Kerr-McGee Section 36

COMPANY: Kerr-McGee Corporation

LOCATION: T14N, R9W, Section 36, McKinley County, New Mexico

OPERATING STATUS: Active as of 2/83, but at reduced production levels[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 reported company mined ore processed at Ambrosia Lake mill was 1,170,473 tons (includes all Ambrosia Lake Mines and two Church Rock Mines) (Kerr-McGee Corp 1982).

CUMULATIVE PRODUCTION:
- The Ambrosia Lake mill, in 24 years, has processed 33 million tons of ore containing 125 million lb U_3O_8 (Kerr-McGee Corp. 1982).
- U.S. Department of Energy production class of 2-20 million lb U_3O_8[b]
- 745,074 tons ore yielding 6,046,780 lb U_3O_8 (0.41%) until 1970.[b]

RESERVES: At 5 operating mines in 1982 (Section 19, 30, 30W, 35, 36) 3,925,000 tons ore containing 17,105,000 lb U_3O_8 (Kerr-McGee Corp. 1982)

DEPTH:
- 1,418 ft (Energy and Minerals Dept. 1982)
- 1,497 ft[b]

GRADE: 0.22% U_3O_8 (average for five mines this study) (Kerr-McGee Corp. 1982).

AGE OF MINE: Began production 1960[b]

OTHER COMMENTS:
- The mine has one vertical shaft, and four exhaust fans with air discharge of 190,900 (ACFM) (Perkins 1979).
- At the Kerr-McGee's Ambrosia Lake properties, approximately 12% of the U_3O_8 is held by mining claims on Federal lands which are owned or leased, while the remainder of the U_3O_8 is held under mining leases on fee lands (Kerr-McGee Corp. 1982). This mine was originally called the Cliffside Mine.

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.
(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Homestake Section 23

COMPANY: Homestake Mining Company

LOCATION: T14N, R10W, Section 23, McKinley County, New Mexico

OPERATING STATUS: Active as of 2/83, but at reduced production levels[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION:
- 1982 - ore from Grants Mines declined to 240,002 tons from 377,537 tons ore in 1981. U_3O_8 produced equaled 1,252,427 lb (Homestake Mining Co. 1982).
- Most of the 1982 ore probably came from the Section 23 Mine, since Section 13, 15, 32 Mines were closed as of 12/31/81 and Section 25 closed since 1/1/82 (Energy and Minerals Dept. 1982). Homestake states that the higher grade areas of Section 23 Mine are now being processed.

CUMULATIVE PRODUCTION:
- 2,528,797 tons ore yielding 9,679,773 lb U_3O_8 (0.19%) until 1970.[b]
- U. S. Department of Energy production class of 2-20 million lb U_3O_8.[b]

RESERVES: Of the New Mexico Mines (stated as three underground mines, which includes Section 23) as of 12/31/82, aggregated 2,150,000 tons ore (proven and probable) (Homestake Mining Co. 1982).

DEPTH:
- 651-783 ft (Energy and Minerals Dept. 1982)
- 830 ft[b]

GRADE: 0.169% U_3O_8 (Homestake Mining Co. 1982)

AGE OF MINE: Began operation 1959[b]

OTHER COMMENTS: The Homestake Grants operations consist of a mill and three underground mines, only one of which is now operating (Section 23). Mineral properties consist of about 3,840 acres throughout the Ambrosia lake district.
Of this total, 2,560 acres are leased from a railroad company, while 1,280 acres are leased from the state of New Mexico. The numerous uranium ore bodies range from 500-800 ft deep, a few to 100 ft wide, up to several 100 ft long and 5-35 ft thick (Homestake Mining Co. 1982).

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.
(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Homestake Section 25

COMPANY: Homestake Mining Company

LOCATION: T14N, R10W, Section 25, McKinley County, New Mexico

OPERATING STATUS:
- Closed since 1/1/82 (Energy and Minerals Dept. 1982)
- Still closed as of 2/83 with vent fans running as needed[a]

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1982 - ore from Grants Mines declined to 240,002 tons from 377,537 tons ore 1981 (Homestake Mining Co. 1982). Since the Section 25 Mine was closed during 1982, probably only a very small percentage of this 1982 total represent the Section 25 Mine.

CUMULATIVE PRODUCTION:
- 1,791,048 tons ore yielding 6,444,889 lb U_3O_8 (0.18%) until 1970.[b]
- U.S. Department of Energy production class of 2-20 million lb U_3O_8.[b]

RESERVES: Of New Mexico Mines (stated as three underground mines) as of 12/31/82 aggregated, 2,150,000 tones ore (proven and probable) (Homestake Mining Co. 1982).

DEPTH:
- 642-801 ft (Energy and Minerals Dept. 1982)
- 811 ft[b]

GRADE: 0.169% U_3O_8 (Homestake Mining Co. 1982)

AGE OF MINE: Began operation 1959[b]

OTHER COMMENTS: The Homestake Grants operations consist of a mill and three underground mines, only one of which is now operating (Section 23). Mineral properties consist of about 3,840 acres throughout the Ambrosia lake district. Of this total, 2,560 acres are leased from a railroad company, while 1,280 acres are leased from the state of New Mexico. The numerous uranium ore bodies range from 500-800 ft deep, a few to 100 ft wide, up to several 100 ft long and 5-35 ft thick (Homestake Mining Co. 1982).

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.

(b) Information from V. T. McLemore, Uranium Geologist, New Mexico Bureau of Mines and Minerals Technology, Socorro, New Mexico.

MINE NAME: Nose Rock

COMPANY: Phillips Petroleum Company

LOCATION: T19N, R11W, Section 31, McKinley County, New Mexico

OPERATING STATUS:
- Placed on standby in 1981 (Phillips Petroleum Co. 1982)
- Still inactive as of 2/83[a]
- Shaft cemented, mine flooded up to certain levels.[b]

DESIGNED MINE CAPACITY:
- Expected production of 2,100 tons ore/day (Perkins 1979)
- Expected full production of 1,000 tons U_3O_8 annually (San Juan Basin Regional Uranium Study 1980).

ANNUAL PRODUCTION: None to date[b]

CUMULATIVE PRODUCTION: N.A.

RESERVES: 25 million lb U_3O_8[b] (Perkins 1979)

DEPTH: Expected depth 3,400 ft (Perkins 1979)

GRADE: 0.14% U_3O_8[b] (Perkins 1979)

AGE OF MINE: Construction stopped in 1981; not yet fully operational, full production was expected in 1982-1983, but now on a "hold" position.[b]

OTHER COMMENTS: The mine has an expected 20 year life; the expected mining recovery efficiency is 95% of the ore in place (Perkins 1979).

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.
(b) Information from C. Wentz, Uranium Specialist, Energy and Minerals Department, Santa Fe, New Mexico.

MINE NAME: Mariano Lake

COMPANY: Gulf Mineral Resources

LOCATION: T15N, R14W Section 12, McKinley County, New Mexico

OPERATING STATUS:
- Closed April, 1982 (Gulf Oil Corp. 1982)
- Still closed as of 2/83, reported that mine is flooded, shaft being filled and surface being restored.[a]

DESIGNED MINE CAPACITY: 625 tons ore/day (Perkins 1979)

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: $\geq$750,000 tons (Perkins 1979)

RESERVES:
- Now depleted (Gulf Oil Corp. 1982)
- Reserves before mining at 3.5 million lb U_3O_8 (Perkins 1979)

DEPTH: 510 ft (Energy and Minerals Dept 1982)

GRADE: 0.24% U_3O_8 with 0.10% U_3O_8 cutoff (Perkins 1979).

AGE OF MINE: Began production 1977, closed (depleted) April, 1982 (Gulf Oil Corp. 1982).

OTHER COMMENTS: The mine had one vertical shaft, and one exhaust fan. With an air discharge of 83,000 (ACFM) (Perkins 1979). The mine area is presently under reclamation.

(a) Information from T. G. Brough, Ph.D., Environmental Scientist, Radiation Protection Bureau, Environmental Improvement Division, Milan, New Mexico.

MINE NAME: Velvet

COMPANY: Atlas Corporation, Minerals Division

LOCATION: T31S, R25E, SW 1/4 Section 3, San Juan County, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: N.A.

RESERVES: As of 6/30/82, Atlas' interests in proven ore-bearing properties approximates 2,172,000 tons reserves (Atlas Corp. 1982)

DEPTH: N.A.

GRADE: Ranges from 0.08-0.52% of economically recoverable U_3O_8 (Atlas Corp. 1982)

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Tony M (Lucky Strike)

COMPANY: Plateau Resources, Ltd.

LOCATION: T35S, R11E, Section 16 and 21, Garfield County, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1980-actual ore tonnage of 34,159 tons (E&MJ, 1981)

CUMULATIVE PRODUCTION: N.A.

RESERVES: 10.9 million lb U_3O_8 under development (average grade 0.15 + %), 6.3 million lb U_3O_8 (average grade 0.15 + %) undeveloped (represents all Plateau Resources properties) (Consumers Power Co., 1982)

DEPTH: N.A.

GRADE: 0.05-0.15% U_3O_8 - represents a number of small deposits in SE Utah held by Plateau Resources (Steyn, 1981)

AGE OF MINE: N.A.

OTHER COMMENTS: Plateau owns or has under lease or option 35,442 acres in Utah. These acres are mostly unpatented and nonproducing claims, with a small part being state leased and fee lands. Property holdings were reduced in 1981, (from 77,500 in 1980) to concentrate on activities in the Henry Mountains, Garfield, Co., Utah (Tony M Mine Area). The NRC issued Plateau a license on 9/21/79 for a uranium processing facility near Shootaring Canyon. This unit will process ore mined from Plateau's properties. Work was suspended as of 3/7/81 because of disputes with a construction company, Mountain States Mineral Enterprises, Inc. Construction was completed in the second quarter, 1982, under Plateau's supervision. Operations commenced June, 1982 and produced 27,267 lb of uranium concentrates in 1982. A decision was made to defer deliveries to the parent (Consumers Power) until 1985, resulting in temporary suspension of facility operation in August, 1982, along with a significant reduction in mining and processing staff levels. The amended contract with Consumers Power called for delivery of 9,100,000 lbs contained U_3O_8 in concentrates over a 13 year period. Consumers Power has a net investment in Plateau of $73.9 million as of 12/31/82 (Consumer Power Co., 1982).

MINE NAME: Wilson-Silverbell

COMPANY: Union Carbide Corporation

LOCATION: T32S, R26E, SW 1/4, Section 15, San Juan County, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Lisbon

COMPANY: Rio Algom Corporation

LOCATION: T29S, R24E, SE 1/4, Section 21, San Juan Country, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: (Integrated mine and mill) Mill capacity increased from 500 to 700 tons per day (Rio Algom, Ltd., 1982).

ANNUAL PRODUCTION: 1982 - 123,000 tons of Lisbon ore processed, or 454,000 lb of U_3O_8 in concentrate which was stockpiled. During 1982, mine production was at 50% normal rate as implemented in October, 1981 (Rio Algom, Ltd., 1982).

CUMULATIVE PRODUCTION: N.A.

RESERVES: Aggregated at 388,000 tons ore as of 12/82. This is noted as proven and probable ore with 3 lb U_3O_8 per ton ore cutoff. These aggregated reserves equal 1.2 million lb U_3O_8, recoverable after allowing for milling losses (Rio Algom, Ltd., 1982).

DEPTH: 2,650 ft final depth (Rio Algom, Ltd., 1982)

GRADE: 1.8% U_3O_8 average recovered grade for 198[illegible]. This equals a mine grade of 3.37 lb of U_3O_8 per ton of ore after allowing for losses in mine extraction and dilution (Rio Algom, Ltd., 1982).

AGE OF MINE: Production began 10/1/72 (Rio Algom, Ltd., 1982)

OTHER COMMENTS: The Lisbon Mine has two shafts and is mined by room and pillar operation. Production work force was reduced 50% on 10/81 (Rio Algom, Ltd., 1982).

MINE NAME: La Sal

COMPANY: Union Carbide Corporation

LOCATION: T29S, R24E, W 1/2, Section 1, San Juan County, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Hecla

COMPANY: Union Carbide Corporation

LOCATION: T29S, R24E, N 1/2, Section 6, San Juan County, Utah

OPERATING STATUS: Active as of 11/82 (U.S. Department of Labor, 1982)

DESIGNED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: N.A.

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: N.A.

OTHER COMMENTS:

MINE NAME: Golden Eagle

COMPANY: Silver King Mines, Inc.

LOCATION: T36N, R73W, Sec. 14 Converse County, Wyoming

OPERATING STATUS: under development[a]

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: No production yet, still being developed as of 6/83.[a]

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: N.A.

GRADE: Average ore grade is 0.15% U_3O_8 as reported in thesis work by Ray Harris.[a]

AGE OF MINE: N.A.

OTHER COMMENTS:

- It will be 3-5 years before production begins, depending on the market conditions.[a]
- Silver King Mines has a management contract with the Tenessee Valley Authority (TVA) for the supervision of exploration, development and production on uranium/-vanadium properties in Wyoming (Golden Eagle).

On May 11, 1978, the Company expanded the initial contract with the Tennessee Valley Authority for supervision of continued exploration, mine development, mining and milling operations, and construction of mine surface facilities and a uranium ore mill. This contract, which was for two years and has been extended for two years with a 10 year option to renew, stipulates that Silver King is to receive a base monthly fee of $19,417 plus reimbursement for additional authorized expenditures and increases computed in accordance with fluctuations in the consumer price index from the date of the agreement.

The aforementioned base monthly fee has exceeded $25,000 since March 31, 1980. Further, the Tenessee Valley Authority has agreed to pay additional monthly fees to Silver King currently aggregating $12,000 (Silver King Mines 1982).

(a) Information from Ray Harris, Soft Rock Minerals Geologist, The Geological Survey of Wyoming, Laramie, Wyoming.

MINE NAME: Big Eagle

COMPANY: Pathfinder Mines Corporation

LOCATION: T27N, R92W, Sec. 2 & 11 Fremont County, Wyoming

OPERATING STATUS: active as of 6/83[a]

DESIGNATED MINE CAPACITY: Production capacity stated as 400,000 tons ore per year by 1979 (Goodier 1978).

ANNUAL PRODUCTION: 1982 152,122 tons ore[a]
1981 260,686 tons ore
1980 455,997 tons ore
1979 118,513 tons ore
1978 104,472 tons ore

CUMULATIVE PRODUCTION: 1978-82: 1,091,790 tons ore as of 12/31/82[a]

RESERVES: in 1978, 7 million lb U_3O_8 proven (Goodier 1978).

DEPTH: N.A.

GRADE: 0.15% U_3O_8 (1978) (Steyn 1981)

AGE OF MINE: began operation in 1978[a]

OTHER COMMENTS:

(a) Information from Ray Harris, Soft Rock Minerals Geologist, The Geological Survey of Wyoming, Laramie, Wyoming.

MINE NAME: Sheep Mountain Operations

COMPANY: Western Nuclear, Inc.

LOCATION: T28N, R92W Sec. 22, Fremont County, Wyoming

OPERATING STATUS:
- Inactive as of 6/83[a]
- The mine has been shut down and on standby maintenance since mid-1981 (Phelps Dodge Corp. 1982).

DESIGNATED MINE CAPACITY: N. A.

ANNUAL PRODUCTION: 1983--0[a]
1982--0
1981--119,147 tons ore

CUMULATIVE PRODUCTION: 1955-1983: 4,519,518 tons ore produced from the Sheep Mountain complex, Crooks Gap District.[a] The Sheep Mountain complex, Crooks Gap District, is made up of the following mines:
Sheep Mountain #1
McIntosh
Golden Goose
Seismic
Reserve
Congo

RESERVES: as of 1978, 7.2 million pounds U308 proven. Indicated reserves to last 20 years (Goodier 1978).

DEPTH: N.A.

GRADE:
- a weighted average grade for the production/period 1955-1973 is 0.24% U308[a]
- 1978 average grade reported as 0.12% U308 (Steyn 1981).

AGE OF MINE: Sheep Mountain operations began in 1955[a]

OTHER COMMENTS: Western Nuclears' mines are now on standby status but could be back in operation on very sho t notice if the demand for uranium oxide so warranted (Phelps Dodge Corp. 1982).

(a) Information from Ray Harris, Soft Rock Minerals Geologist, The Geological Survey of Wyoming, Laramie, Wyoming.

MINE NAME: Schwartzwalder

COMPANY: Cotter Corporation, a wholly owned subsidiary of Commonwealth Edison of Chicago.

LOCATION: T2S, R71W, Sec. 25, Jefferson County, Colorado

OPERATING STATUS: Active as of 11/82 (U.S. Dept. of Labor 1982)

DESIGNATED MINE CAPACITY:
- daily ore capacity of 500 tons ore per day (Wright 1980)
- 600 tons ore per day (E & MJ 1981).

ANNUAL PRODUCTION: 1980: 100,000 to 500,000 short tons ore (E & MJ 1981).

CUMULATIVE PRODUCTION: N.A.

RESERVES: N.A.

DEPTH: The mine is much deeper than it is broad in any aerial dimension. It is 2000 ft. deeper than it was when Allen Birds did a thesis study on the mine in 1958 (Wright 1980).

GRADE:
- 0.30% U_3O_8 (E & MJ 1981). 0.20 - 0.25% U_3O_8 with a 0.09% cutoff (Wright 1980)

AGE OF MINE: opened in 1956 (E & MJ 1981).

OTHER COMMENTS: This mine is the premier vein-type uranium deposit. It has three shafts and is mined by open stopping of the vein. By products of mining are molybdenum and silver (0.1-2.0 oz silver/ton ore) (E & MJ 1981).

The Illinois vein is up to 50 ft. wide, with narrower horse-tails averaging 1.5 feet wide. All uranium produced by the mine is committed to use by Commonwealth Edison (Wright 1980).

MINE NAME: Bill Smith

COMPANY: Kerr-McGee Corporation

LOCATION: T36N, R74W Sec. 36, Converse County, Wyoming

OPERATING STATUS: Underground mine closed. Environmental requirements have stopped all but one open pit operation (Steyn 1981).

DESIGNATED MINE CAPACITY: N.A.

ANNUAL PRODUCTION: 1979-83 0[a]
1978 5,482 tons ore
1977 9,098 tons ore

CUMULATIVE PRODUCTION: 1977-1978: 14,580 tons ore.[a]

RESERVES: 60 million lbs. (?) U_3O_8 reserves in-place estimated (Steyn 1981)

DEPTH: N.A.

GRADE: N.A.

AGE OF MINE: began operation in 1977[a]

OTHER COMMENTS: The underground mine went on standby status in 1978, with no production since[a] (Goodier 1978).

(a) Information from Ray Harris, Soft Rock Minerals Geologist, The Geological Survey of Wyoming, Laramie, Wyoming.