United States Department of the Interior

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May 18, 1987

for R. Ken Ballard / Raj Natarajan MS 623SS

Mr. Patrick L. Boileau
Department of Energy
Richland Operations Office
Richland, Washington 99352

Dear Mr. Boileau:

In fulfillment of the Interagency Agreement between our respective government organizations, attached is Task 1: Bureau of Mines Review of Department of Energy Report "Exploratory Shaft Design Basis Study Review."

Due to the short time limitation and the need to have this review conducted by different Bureau of Mines research centers, the review comments were left in their original format.

If you have questions about the review or would like to question the authors in person, please call me at FTS 723-6550.

Sincerely yours,

Roger L. King
Research Supervisor
Ground and Methane Control

Attachments

See ltr to Ballard
fm. Cook 5/18/87
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Task 1: Bureau of Mines Review of Department of Energy Report "Exploratory Shaft Design Basis Study Review"

CONTENTS

Task 1: Bureau of Mines Review of Department of Energy Report "Exploratory Shaft Design Basis Study Review"

PURPOSE

The Basalt Waste Isolation Project (BWIP) is an investigative program being conducted by the U.S. Department of Energy (DOE) to determine the feasibility of locating a nuclear waste repository in the deep basalts beneath the Hanford Site in south-central Washington. As part of the site characterization phase of the program, an Exploratory Shaft Facility (ESF) will be constructed to conduct in situ testing for site characterization and feasibility.

The Bureau of Mines has been solicited by DOE, through an Interagency Agreement, to review the subject report as an independent reviewer.

SCOPE

The review of this report covers three general areas: Hydrology, Ventilation, and Shaft Sinking and Excavation. The Hydrology section evaluates water inflow parameters and quantities and the water inflow range. The Ventilation section evaluates underground ventilation air quantities; methane inflow parameters and quantities; gassy mine cost impacts; shaft ventilation and conveyance velocity criteria; and methane release rate. The Shaft Sinking and Excavation section evaluates the second shaft cost estimate and hoisting; underground development schedule; and underground drifting.

HYDROLOGY

This review pertains to the following sections of the Exploratory Shaft Facility Design Basis Study Report, Section 4.1.2, 4.1.3, Section 5.2, Appendix A, and Appendix A Annex.

In general, I believe that the conceptual model of the hydrologic setting for the ESF, as presented in this report, is legitimate and defensible. The application of analytic models to the various inflow problems is at this stage adequate. The inflow estimates, and ranges are correctly derived and reasonable, given the modeling assumptions, and the limited availability of hydrogeologic field data at this time.

Many of my comments deal with presentation of material in the report, especially from the standpoint of one interested in just the hydrologic setting and the inflow calculations. The organization of this report, i.e. summary first, followed by a more detailed hydrologic analysis including modeling, followed by even more detailed geologic and hydrogeologic description, makes it difficult for the reader who starts at the beginning of the report to follow the development of specific concepts and definitions. In my opinion, if someone were going to read the entire section on hydrology, they should start with pages 69-97, then read pages 34-68, then pages 18-23, and finally pages 8-9.

I also have a few comments about the application of the equivalent porous medium, steady-state analytic model. The application of this analytic solution is adequate, but could have been extended a little further, to look at inflow to drift locations that are closer to the interflow boundaries than 37.5 m, especially since there appears to be a number of instances where this is the case.

. The following pages contain specific comments, identified according to page and paragraph:

Page 8

Inflow Processes

Paragraph 2

At this point in the report, the terms feature and discrete feature are not well defined, from a hydrologic standpoint, and it leads to some confusion in this section. Not until page 21 are these terms adequately defined. In general, it is hard to understand the discussion in section (4.1.3) without first having read appendix A. A lot of this has to do with casual use of terminology that isn't really defined until later.

I think the second paragraph in 4.1.3 might be better written as follows...

....The flow from these discrete features (fractures or aquifers?) will be initially transient. Because of higher piezometric head during the transient phase, inflow will be greater than during the subsequent steady-state phase. A program to restrict (implies grouting) or reduce inflow may be instituted during the transient phase. Such a program will have as its aim, the reduction in transient phase mine inflow to steady state levels.

How are normal operating conditions defined? Hydrologically? Does normal refer to steady-state inflow conditions?

Paragraph 3

I think it would be more precise to say that the purpose of the probe holes is to monitor potential for inflow to the advancing drift (this includes transmissive and piezometric conditions ahead of the drift). It is also a means of reducing piezometric pressure, and therefore dewatering rock in advance of the drift development.

Paragraph 4

The comments about one-dimensional fracture flow and vertical fractures makes absolutely no sense unless you've read appendix A--same for the comment about vertical fractures and steady-state conditions. Sounds as if you are saying that flow in horizontal fractures or horizontal direction is in a perpetual time-dependent state. Again, the comment about two dimensional flow is hard to figure out, unless the reader has really looked carefully at the yet-to-be presented, model assumptions.

Page 9

Paragraph 1

The phrase involving...majority of ground water...sounds vague and uncertain. It might be better to say that it is anticipated that inflow in the vesicular zone would be greater because of the greater transmissivity of this material, relative to the same length of excavation in the interior.

Paragraph 2

These "zones" are really features; this change in terminology plus the ambiguous term "localized features" adds to the confusion. Again, I don't understand the need for the comment about transient flow in this paragraph. If you think it is important to educate the reader regarding transient and steady-state flow, it might be best to make a general comment early-on, about the development of steady-state flow conditions from an initially transient state, in the vicinity of a well or drift.

Page 21-22

Section 5.2.3

The discussion of steady state inflows should follow some prior discussion of transient inflow, so the reader can see the basis for choosing 1 week as the transient interval. Its clear from reading c.b. 0617 that this is a

legitimate number, but it doesn't come across in section 5.2.3. You might want to include some additional transient inflow data from c.b. 0617.

Appendix A

Page 38

Paragraph 4

Small point...., conceptual hydrogeologic models are developed by ASSOCIATING geologic model with hydrologic parameter values (conductivity, etc.).

Page 48

Paragraph 3

I think there is some confusion over fracture terminology. Golder points out (section 5.4.2) that 16 cm fractures are really 16 cm wide zones of fracturing. On page 48, however, this 16 cm zone is referred to as an open fracture. To me, open fracture denotes an unfilled fracture, but this can't be right.

Not until page 87-90 is there some precise definition of fracture terminology. It would help the reader a lot if some definitions for fracture aperture, fracture width, filled fracture, open fracture and zone of fractures were introduced on or before page 44 (in connection with figure A-2 would be a good place).

I think that the measured T value of 9.47 X 10-4 for a fracture (zone) encountered in Umtanum, in the RRL-2 bore hole, is an important number. The K' value of 5.18 X 10-4 of this fracture zone indicates that the assumed width of the zone is 1.83 m. However, the probability encoding indicates fracture (zone) width in the dense interior is in the range of .16 to .55 m. If we use these numbers to calculate K' instead of 1.83 m, then K' for the fracture zone is in the range of 5.9 X 10-3 to 1.72 X 10-3.

The report states that core recovery was not possible in this interval. If there is field evidence that the fracture zone was indeed 1.83 m wide, then this is the best number for calculating K'. However, if this is the case, then how does this number square with the probability analysis which puts the 90% confidence interval on fracture (zone) width at 55 cm? Is the difference due to the presumed angle at which the bore hole intersects the fracture zone? Or, is there some other reason that should be explained to the reader?

I presume that primary cooling joints/fractures (page 35) are the same thing as entablature joints and colonnade joints (figure A-2), which are then represented as a densely fractured medium in Model 1, figure A-5; although to this point, I don't recall that this has been stated explicitly.

Page 52-55

While conceptual models 2, 3, 5 and 6, presented on page 54, are rejected for geologic reasons, models 1 and 4 are rejected (in large part) for hydrologic reasons. It is somewhat confusing to the reader when you say that you reject models 1 and 4 because they don't account for recharge of the Cohassett, when the models are presented (page 52) as simply "descriptions of the basic fracture network" within the Cohassett flow interior. From your discussion on page 55, they appear to be adequate models of the geologic setting, but the hydrogeologic setting (naturally) must be broader, to include layers which supply ground water recharge to the Cohassett. I think rephrasing the discussion on the first half of page 55 would help.

Page 56

First sentence doesn't make sense--some words or a line evidently missing.

Page 58

Paragraph 2

Since you are now talking about MODELING the interior, I think that you should clearly state that the primary cooling fractures in the interior are dense enough such that the interior can be modeled as an equivalent porous medium (EPM), with a hydraulic conductivity of 1.0×10^{-13}. I don't think the term equivalent hydraulic conductivity is used correctly here.

Also in this paragraph, the sentences dealing with transient and steady-state inflow are extremely vague. They suggest to the reader that the time required for steady-state inflow conditions to develop in the EPM rock is unknown, and possibly very long; especially so, since there is no transient flow analysis for the EPM rock in this report or in computational brief 0617. It's not clear whether this is because the specific storage of rock in the flow interior is insignificant, or because the geometry of the drift precludes a simple analytic model for transient inflow. You may want to clarify this, or else simply delete mention of EPM transient flow in the paragraph.

Page 59

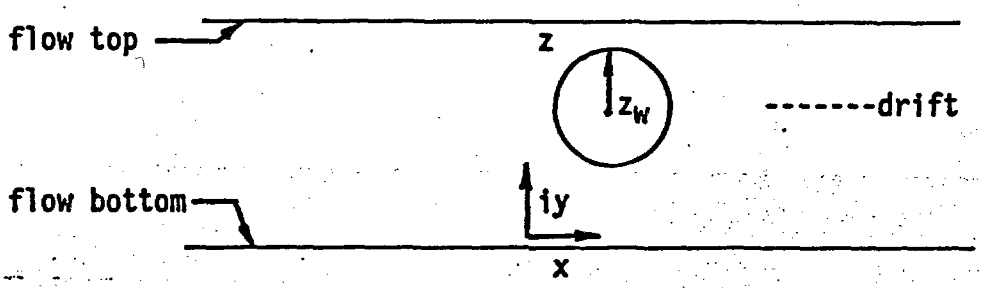
The reference given for the steady-state inflow equation on page 59 of the report is obscure, and the solution presented may be unfamiliar to many of the readers. Unlike the other conceptual models presented subsequently, it is much more difficult for the reader to verify that the equation is correct, and that the boundary conditions imposed on this solution are appropriate to the current problem. I think it is especially important therefore that the assumptions presented for this model be formalized as hydrologic boundary conditions. It would also be helpful to include a few sentences describing how the solution is derived, or at least an additional reference source.

I was able to find an analytic solution to the same problem, derived using a conformal mapping technique involving Schwartz-Christoffel transformations. The solution is in a book entitled, "Groundwater Mechanics" by Otto D.L. Strack, Prentice Hall 1987, pages 350-352. Since I have the book in manuscript form, I am not sure if these page numbers are still correct; however, it is in a section entitled, "Applications of Conformal Mapping." The expression (on the next page) is developed as an exact solution for flow to a well that is located midway between two infinitely long, parallel, constant head boundaries.

This solution is more general than the one presented in your report and would permit specification of the drift at different elevations relative to the interflow boundaries. It can also be modified to permit a difference in head boundary conditions above and below the drift, or a uniform ground water flow gradient through the flow interior. (You indicated earlier that there was an upward vertical flow gradient). If you assume the drift (well) is located exactly midway between the two boundaries, I have verified that this solution reduces to the one presented on page 59 of the report.

Expression for parallel constant head boundaries:
(from ODL Strack, 1987, page 352)

$$-Q = 2\pi Kb.\Delta h.\ln\left[\frac{e^{\pi z/d} - e^{\pi \bar{z}_w/d}}{e^{\pi z/d} - e^{\pi z_w/d}}\right]$$



where $z_w = x_w + iy_w$ = center of the drift
$z = x_w + i(y_w + r_w)$ = a point on the perimeter of the drift.

Radial constant head boundary:

$$Kbh = \frac{Q}{2\pi} \ln(r) + e$$

when $r = r_w$, $h = 0$
when $r = {}^{d}/_{2}$, $h = 900$

$$-Q = 2\pi.\ Kbh\ \ \ln\left[\frac{r_w}{d/2}\right]$$

$-Q = 3.99 \times 10^{-7}\ m^3/s$ [compare to $3.7 \times 10^{-7} m^3/s$ from table 4.2]

Along the same line, I think it is worth noting that if you solve this flow problem with a slightly simpler boundary condition, you get almost the same result. On the next page I solved the inflow problem using your numbers for the pre-connect phase of inflow, and a much simpler analytic solution for a very large diameter well. The only difference in boundary conditions between the two solutions is that the simpler expression assumes a radial, constant head boundary a distance d/2 away from the center of the well, instead of two parallel boundaries a distance d/2 away from the well. As you can see by comparing these numbers to those in table 4.2 of brief 0617, the difference in inflow is insignificant, given the specified parameter values.

The parallel boundary conditions would be more important, and inflow to the workings would be greater, of course, if the drift were constructed closer to the interflow zones than 37.5 m. For instance, figure 4 of brief 0617 shows some workings constructed within 10.1 m of the flow top. Also, if the vesicular zone does have a permeability that is 2 to 5 orders of magnitude greater then the flow interior, (table 5.3, brief 0617) then it is reasonable to calculate inflow to the main drift, (in at least one scenario) under the assumption that, prior to developing the raise, the lower boundary of the vesicular zone is a constant head condition. In this case the main drift would be 17.1 m from the upper flow boundary and 27.7 m from the lower boundary. Inflow to the main drift under these conditions would be greater, than what has been estimated, although it is not clear whether it would be significantly greater. In any case, it could be calculated relatively easily using the analytic formulation developed by Strack.

...also on page 59:

There is a typo in the expression for sv. The inverse hyperbolic tangent function should be written as...arctanh [x]... Also in this expression, rw is drift radius (assumed to be circular), not drift width.

Since the first two assumptions regarding the use of this model are important hydrologic boundary conditions, I think they should be stated explicitly (at least parenthetically) below each assumption:

assumption 1. (A constant head exists at the boundary between the flow interior and the interflow zones above and below the drift.)

assumption 1a. The drift is assumed to be developed exactly and entirely midway between the flow top and flow bottom.

assumption 2. (The drift is bounded by impermeable layers at both ends.)

Also, I think assumption 4 should include specific mention of the vesicular zone as being part of the flow interior that is assumed homogeneous and isotropic, according to this model.

Page 61...

Paragraph 2

I was unable to obtain the Lohman paper, and there are some things about the explanation of the model that are unclear. Brief 0617 says nothing about an assumption of one dimensional flow for this model. As I understand it, in the plan view, the line sink is a one dimensional entity, with a vertical length that is equal to the vesicular zone thickness, and a horizontal length of (1). Multiple line sinks are used to model the drift. Thus, in the plan view, flow to the line sink is two dimensional, not one dimensional. Is this correct, or am I missing something?

Also a typo on this page: the radicals are missing from the equation for Q(t) on this page.

Section A.4.2 Numerical Models

Paragraph 3

I think it is an oversight not to include semi-analytic methods in the category of models that are capable of handling the more complex flow problems not dealt with in this report. The boundary integral element methods (BIEM),

for instance, is a particularly powerful semi-analytic method for dealing with problems of flow in layered, heterogeneous or fractured rock. In fact, semi-analytic BIEM methods are far more efficient in dealing with flow problems that involve discrete fractures in a permeable rock matrix than are standard numerical methods.

The model presented in PNL-4005/UC-70, "Analytic Modeling of Flow in a Permeable Fissured Medium," (ODL Strack, 1982) is one example of the use of semi-analytic BIEM methods for these problems.

Page 63...

Paragraph 2

A specific recommendation that I think should be incorporated in the next stage of modeling:

Use hydrologic modeling to assess inflow rates at specific locations in the drift, where the proximity of the workings to flow features such as an interflow zone, or the vesicular zone, are significantly different from the average value assumed thus far by the analytic models.

VENTILATION

"The one parameter which is common to all questions about design performance confirmation is the ventilation required to provide a suitable underground environment." (page 100). The ventilation plan selected involves:

1. Intake air velocity within ES II.
2. Methane dilution.
3. Gassy mine regulations
4. Heat and humidity control
5. Airborne contaminant control

The following comments discuss how the design basis and conceptual design studies address each of the above.

1. INTAKE AIR VELOCITY

The ESF (Exploratory Shaft Facility) design requires that the inside diameter of ES I be six feet. Evaluation of shaft size was based on the requirements for ventilation during various phases of underground development and testing. In addition, the DOE requires that the air velocity in an ESF shaft used to convey personnel not exceed 2,000 fpm. To provide adequate airflow underground without exceeding the DOE maximum intake shaft velocity, the proposed inside diameter of the second shaft, ES II, was set at 10 feet. The "key recommendation" identified in the design basis flexibility study is that ES II will be the intake shaft and ES I will be the exhaust shaft. The decision to provide a 10 foot diameter shaft for intake ventilation seems reasonable in light of ventilation requirements.

2. METHANE DILUTION

A primary objective of the "Exploratory Shaft Facility Flexibility Study" was to determine if the ventilation system was adequate to dilute the methane gas entering the mine and thus prevent a methane ignition. The quantity of methane liberated and the amount of intake air needed to dilute it will vary with the stage of facility development. The ventilation system must provide adequate methane dilution from the time underground construction begins until repository construction is authorized. Five phases or scenarios in ESF life are summarized on pages 10 and 11. Due to the nature of methane control, major ventilation flexibility will be required during scenarios 1 and 2.

During construction of the ESF, methane release rates are likely to be the highest and frequent modifications of the ventilation system will be needed to assure methane dilution. The airflow quantity and techniques for airflow control established during ESF construction should be adequate for methane control after completion of construction up to the time that the ESF is connected with the repository.

Quantity and Location of Methane Inflow

Methane has been found dissolved in groundwater within the basalt. No other source of methane has been identified. Estimates of methane inflow assume that the only source of methane is the gas that is dissolved in groundwater. If this is true, it may be inferred that the quantity of methane encountered in the ESF will be directly related to the groundwater inflow rates and the quantity of methane dissolved in the groundwater.

The rate at which methane will flow into the ESF is difficult to estimate without underground or borehole data. Information from the five boreholes drilled at the location of the proposed repository were considered suspect because of sampling errors (cf. page 9).

The maximum amount of methane dissolved in one liter of groundwater entering the ESF was estimated to be 1,050 mg. At atmospheric conditions in the mine this would convert to 0.052 ft^3 if 100 percent of the gas is liberated from the water.

These estimates of inflow rate were accepted as reasonable and used to evaluate whether airflow quantities were adequate for methane dilution.

In addition to the quantity of methane dissolved in the groundwater, water inflow characteristics will determine at what rate methane will enter the ESF. The uniform fracture characteristics of the basalt suggest that any water present will enter uniformly throughout the mine workings. Initially this release rate will be steady and potentially the total quantity of methane released in the ESF could increase with time. However, this "steady state" inflow will persist for approximately one week following intersection. After this the groundwater inflow rate will decrease significantly with time. The potential for the highest rate of methane release will occur immediately after new rock is exposed.

Control of Airborne Methane

The conceptual design study suggests a main airflow through the ESF of 50,000 cfm. Increasing the diameter of ES II allows an airflow of 79,000 in the underground workings without exceeding a velocity of 2,000 fpm in the shaft. The increased air quantity should provide adequate airflow for diluting the total quantity of methane expected to be released in the ESF. However, the design basis study points out that methane liberation will probably not be uniform throughout the ESF. As noted above, the rate of liberation will be greatest at the faces where mining is taking place. The worst case situation (the maximum concentration of methane generated) would occur if all water inflow occurred in one entry rather than uniformly throughout the ESF. Given a maximum water inflow of 63.73 gal/min, an air quantity of 5,000 cfm would be adequate to maintain the methane level below 0.25%. The design basis study does not show how fresh air will be distributed through the ESF. Based on the information provided, the total ventilation quantity, 79,000 cfm, should be adequate for methane control if it is properly distributed throughout the mine entries.

The affect of the ventilation on methane dilution can be assessed after the ESF is in operation. Tracer gas techniques used by the Bureau of Mines are recommended (1, 2, 3, 4)[1]. The technique, which uses SF_6 as the tracer gas, is useful in evaluating the effectiveness of auxiliary fans, and probing the air circulating near a working face region where ventilation appears to be poor.

Current MSHA regulations (57.21034) for mines classified as gassy require that "The quantity of air coursed through the last open crosscut in pairs or

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[1]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

sets of entries or through other ventilation openings nearest the face, shall be at least 6,000 cubic feet per minute..." In addition, proposed MSHA regulations for gassy mines (57.36210) add: "The quantity of air across each face at a working place shall be at least 2,000 cubic feet per minute."

Very limited information is given in the design basis report concerning how airflow will be directed from the last open cross cut to the face. The conceptual design report does describe ventilation tubing size and placement. However, a key recommendation included in the design basis study is to reverse the planned airflow direction. It is unclear how this airflow reversal will affect the face ventilation plan, but it is assumed it will have no impact prior to initiating ventilation between the two shafts. The effect of ventilation flow direction on face ventilation should be addressed in the definitive design study.

To comply with the MSHA regulations, at least 6,000 cfm of air will have to be directed through the tubing in each entry. To determine compliance with this standard, airflow quantity would be measured at the end of the tubing closest to the face whether blowing or exhausting ventilation was used. The conceptual design study shows that blowing ventilation will be used before airflow is established between ES I and ES II. After main airflow is established, all face ventilation will be exhausting.

Studies to compare the effects of using exhaust versus blowing ventilation (5, 6) have shown that significantly different face ventilation airflow patterns occur. Blowing ventilation is more effective for distributing air to the face. For the same airflow quantity, the range of influence for blowing airflow is much greater than for exhausting airflow. This is particularly important when considering the dilution of methane. Moreover, it is easier to provide 2,000 cfm across the face if blowing ventilation is used. To attain the same 2,000 cfm at the face with exhaust ventilation, the tubing would have

to be kept closer to the face. The location of the exhaust tubing near the face can interfere with mining operations at the face and it is more subject to damage. Also, the tubing must be advanced more frequently to maintain an effective airflow across the face. A disadvantage to using blowing ventilation is entrainment of particles due to the relatively high air velocities. Particle entrainment is discussed later in these comments.

Several techniques have been tested (7, 2) to improve airflow distribution at the face while using exhaust ventilation. When using these techniques care must be taken to prevent recirculation of air which can increase methane concentration. (57.36209)

Use of ducting is the most efficient way to separate intake and return airflow in the working entries. However, in the event of a power failure or fan failure, some alternative method of ventilating the entries must be provided. Curtains and regulators in the main airflow entry are shown in the conceptual design report. Some explanation is needed to describe how these devices will be used to control face airflow.

Standard for Airborne Methane Levels

Shown on table 5-3 are the ventilation quantities of fresh air required to dilute the methane level to below a 0.25% concentration. On page 23 it is stated that this level (0.25%) is "...required by the Mine Safety and Health Administration." It should be noted that the 0.25% figure was established for designating gassy mines [57.21001 (c)] and not as a safe operating level (See discussion of MSHA gassy mine regulations below). According to current MSHA gassy mine regulations, mining can be conducted as long as the methane concentration does not exceed 1% (cf. 57.21039).

If DOE does not feel the MSHA methane standard is appropriate, a permissible level of methane below 1% should be proposed. At present the design basis for methane control depends on the best estimates of the worst case conditions (i.e. maximum water inflow rate, and 100% liberation of methane from water). The actual methane liberation rates and distribution of methane throughout the ESF will not be known until after mining begins.

Continuous Methane Monitoring

A permissible standard for methane is only as good as the technique used to monitor its concentration. The definitive design for the ESF should include a comprehensive methane monitoring plan that will characterize the methane release rates and the adequacy of the ventilation provided.

Weekly monitoring for methane is specified in the old (57.21056) and proposed MSHA regulations (57.36211). If continuous mining machines are used at the face, continuous monitoring must be conducted with permissible monitors mounted on the mining machines and placed as close to the face as practical. Whether or not continuous mining machines are used, continuous monitors at appropriate locations provide one of the most effective ways to assure that levels of methane do not exceed permissible levels. Continuous monitoring systems that can provide immediate sampling results from selected remote locations, as well as sampling procedures for their use, have been evaluated by the Bureau of Mines. (8, 9, 10, 11).

Additional studies are needed to determine if there are other sources of methane. Probe holes drilled in advance of mining could be used to monitor the occurrence of methane in advance of mining or, if necessary, drain off methane before it enters the mining environment. These borehole measurements could also identify if there were sources of methane other than groundwater,

in advance of mining. Work by the Bureau of Mines has identified techniques to remove methane using horizontal holes drilled from active workings of coal mines (12, 13, 14).

3. GASSY MINES REGULATIONS

The impacts of six alternative designs for complying with MSHA gassy mines standards are presented. These are summarized on page 130 of the design basis study. A seventh alternative assumes the site is classified gassy after the construction begins and, subsequently, changes must be made to meet the gassy mine requirements. As noted in the design basis report, the seventh alternative would result in substantial cost and schedule impacts caused by delays and retrofitting. Therefore, this is not considered an acceptable alternative. It should be assumed that the mine facility will be designated gassy before site preparation begins.

The remaining six designs involve compliance with either existing or proposed MSHA gassy mines regulations. It will also be required that the ESF comply with mining laws for the state of California. The Bureau was unable to obtain a recent copy of these mining laws and therefore no further comment is made concerning their application to the ESF mining site. It is possible that MSHA will have no jurisdiction to enforce their regulations at the ESF, but DOE will act to assure that the site contractor does comply with the regulations.

The worst case situation (viz a viz no variances granted) would require compliance with one of the following two alternatives.

1. Category III with no variances
2. Current MSHA regulations with no variance

It is recommended that emphasis be placed on designing the ESF to comply

4.. HEAT AND HUMIDITY CONTROL

"One of the primary criterion used to determine the shaft size for an underground facility is the requirement for ventilation (p. 5)" One of the design criteria for the ESF is that air temperatures underground shall not exceed 80 degrees at the work places of personnel. Water temperature at the ESF is about 125 degrees and, if there is significant inflow, it will significantly impact on heat and humidity load.

A computer program was used to estimate ventilation requirements for heat and humidity control. Based on this work, it was concluded that "..the ventilation and air conditioning system can support the scheduled activities in the ESF with the maximum water inflow under the worst conditions without additional cooling." (p. 27). However, it is assumed that the intake air will be bulk-cooled on the surface to 50 degrees F. Little information is given about the air conditioning system except that the cooled air will be released below the collar of the intake shaft.

5. AIRBORNE CONTAMINANTS CONTROL

On page 115, it is noted that "The environmental conditions at the various work areas were modeled to comply with threshold value limits recommended by the American Conference of Governmental Industrial Hygienists (ACGIH)." A list of contaminants that may be encountered should be provided. Due to different TLV's, different contaminants may require different ventilation quantities, or dust control techniques, to maintain compliance.

Before flow-through ventilation is established between the two shafts, blowing face ventilation will be used. Air velocities required for methane control may entrain settled dust and increase airborne levels of these contaminants. Some auxiliary technique for dust control, such as a scrubber may be required for use with the blowing system.

. The exhaust ventilation system is more effective for control of airborne contaminants, but the tubing inlet must be kept close enough to the source of the dust to provide necessary capture velocity.

Sprayed water is one of the most effective techniques for suppressing airborne dust (15). If ventilation is not adequate to control airborne particles, techniques for using water should be evaluated. Specially designed water sprays are also effective air movers which can be used to improve airflow near the working face. Certain mining applications have used water sprays to control dust and dilute methane.

The following are recommended changes or additions for definitive design study:

1. The face ventilation system is important in the control of methane, particularly in the vicinity of newly exposed rock.

- A detailed description of the face ventilation system should be given showing the main ventilation airflow from ES II to ES I.

After flow-through ventilation is established, exhaust ventilation will be used in each heading.

- Indicate how auxiliary systems, such as auxiliary fans and diffusers will be used to improve face airflow.

When the operation of the face fans is disrupted, or there is a prolonged period of inactivity in the mine,

- Indicate how positive face ventilation will be maintained using regulators, curtains etc.

2. Accepting estimates of methane inflow rate, the ventilation plan presented in the design basis study is adequate for reducing the methane levels well below 1%. However, additional data is needed to confirm methane release rates and the adequacy of the ventilation system for all operating conditions. Therefore, it is recommended:

. - A mine wide monitoring system be developed. This system should be capable of remotely monitoring methane levels at each working face allowing correlation of these levels with the airflow supply in an entry.

- A technique for monitoring methane released from the horizontal boreholes should be prepared.

3.- Provide additional information about the distribution system for the bulk cooling system.

4. - Prepare a list of potential airborne particulates which could be harmful to the workers' health if not controlled in the ESF.

SHAFT SINKING AND EXCAVATION

Second Shaft Size

The objective of the Second Shaft (ES-II) Scoping Study was to determine the optimum shaft diameter for maximizing the ventilation and hoisting capacities. The shaft diameters to be considered are: 6, 8, 10, and 12-ft drilled, steel-lined and grouted in-place finished inner diameter.

ES-II will be used as the air intake shaft because it has previously been determined by DOE that ES-I will be limited to a 6-ft diameter shaft. The rationale for this decision was based on the following for ES-I: slow hoisting speed (750 ft/min), limited hoisting capacity (20 tons/h), low air volume dictated by ES-I shaft size (6 ft), and limited ventilation capabilities (2,000 ft/min air velocity).

Technical considerations relating to sizing an air shaft indicate that the best design would be a circular, lined shaft. The circular shaft distributes ground stresses most favorably, provides the greatest working area for its perimeter (effecting reductions in lining material costs and rubbing surface for the air), and is the cheapest to sink. Every attempt should be made to

site the shaft so as to reduce the impact of undesirable geotechnical features within the constraints imposed by the nuclear waste repository.

Rotary drilling offers distinct advantages in terms of safety, minimum construction duration and least damage to the rock and groundwater regime. It is considerably more expensive than all other methods; however, this disadvantage is often more than compensated for by the considerable savings in capitalized costs for the repository project which accrue from the short construction time. Moreover, rotary drilling is especially suited for unfavorable geological conditions such as indicated at this project site.

The primary function of the ES-II shaft will be for ventilation of the underground workings. Mine ventilation is required to dilute, render harmless, and carry away dangerous accumulations of gas, dust and heat from the working environment. Therefore, the size of ES-II will depend on the ventilation requirement and the shaft size will also determine the shaft cost.

The volume of ventilation required will be determined by the maximum demand which will be during construction of the underground facility after connection to the ES-II shaft and during the initial repository construction. During both these cases, multiple headings will be driven concurrently resulting in the maximum heat loads on the ventilation.

The subject report contains table 4-3 which reflects the different shaft sizes and their respective construction time. It can be seen from table 4-3 that the construction duration is relatively insensitive to the shaft diameter. For example, more than doubling the shaft ventilation capacity (increase shaft ID from 6 to 10 ft) results only in a 20% increase in construction time.

The variations of construction costs with shaft diameter, as shown in table 4-3, follow a similar path. More than doubling the ventilation capacity

(increase shaft diameter from 6 to 10 ft) raises the construction cost only 30%.

The airflow required to ventilate the proposed ES layout and initial repository construction was determined to be 96,300 ft^3/min. A 10 ft diameter shaft will generate an airflow of 96,840 ft^3/min, as indicated in the report. The report also specifies that the air velocity through the unobstructed area of ES-II will be 1,153 ft/min, and, through the unobstructed area of ES-I will be 3,000 ft/min. Therefore, a shaft with an ID of 10 ft provides adequate ventilation to support the ESF construction and testing and the initial repository construction and is relatively insensitive to cost and construction time.

Second Shaft Cost Estimate

Construction methods have been changing in recent years for shaft sinking from labor intensive conventional methods to mechanizing the construction process which requires more geological and geotechnical data to avoid costly delays. When devising the Exploratory Shaft Facility, it is essential to correlate the actual investigations to the problem areas of construction, as well as to provide the basis for design assumptions and engineering cost estimates. If all the potential problem areas are detected and the possible consequences recognized at an early stage in the design/construct process, all necessary design and construction activities can be geared to overcome them.

In recent years there have been very few projects carried out for special purposes, and therefore case history data is limited. The only shaft data of comparable size was for an exploratory shaft in Carlsbad, New Mexico (16). This was an 11 ft 10 in diameter shaft drilled to a depth of 2,272 ft at a total cost of $10,361,071; only the top 850 ft of shaft had a steel liner.

The total estimated cost for ES-II, assuming a 10 ft shaft, is $23,439,000, as shown in table D-10 of the subject report. However, every shaft is unique and it is very difficult to compare costs because of the uniqueness of the geology and site conditions.

Shaft Hoisting Requirements

The major design of this hoist system is to transport personnel, equipment, tools and excavation rock from the work area to the surface. Its capacity must be such that it will meet predetermined production schedules as well as anticipated future production needs. Since a shaft often provides the most direct access over the longest period of time, there is an advantage to designing a shaft for maximum duty. The current trend in shaft design is to provide multipurpose shafts for handling excavation, materials, personnel, services, manways, and ventilation. The shaft's ultimate requirements must be defined during the shaft size selection phase, because once a shaft is excavated and equipped, it cannot be easily enlarged in the future. An examination of Appendix B (Table B-1) indicates that the maximum amount of material expected to be hoisted is 3099.69 tons per week. Since any one of the shaft sizes shown in table 4-3 would meet this goal, the hoist requirements are not dependent on shaft size.

Underground Development

The underground facility will be constructed in basalt which has compressive strengths ranging from 40,000 to 60,000 psi. The high compressive strength of basalt, plus the relative short length of drifts, limits the type of methods that could be used for excavation. The drill and blast method with controlled blasting is the only proven method. A tunnel boring machine would be uneconomical. The only other known mechanized method of excavation that

would give a smoothly bored rectangular heading and penetrate the hard rock would be the Mobile Miner. This machine was recently introduced by the Robbins Company and uses disk cutters similar to ones used on a tunnel boring machine. This machine would elimate the overbreak caused from the controlled blasting, allowing the host rock to remain undisturbed. Other improvements include improved air flow, minimal roof support and a flat invert.

After ES-I shaft completion, the construction is very low due to the limited space and ventilation capacity and the need to proceed cautiously to drive openings with the least impact to the host rock. In fact, the maximum weekly tonnage is only 3.099.69 tons/week, which is very conservative when considering that average conventional mining tonnage is 600 tons/shift. The rate of advancement, which is not production oriented, is well within conventional mining practice.

Recommendations/Questions:

If ES-I was not predetermined by DOE to be a 6 ft diameter shaft, there would be many reasons to reconsider enlarging ES-1 to a 10 ft shaft: low air volume, limited hoisting capacity, limited ventilation capacity, and the capacity to bring in mechanized equipment. However, could ES-I be a 10 ft shaft, leaving ES-II as the 6 ft shaft?

2. Could mechanized equipment, such as the Mobile Miner, be used on a trial basis to determine its feasibility for driving entries? If successful, this machine could preserve the host rock in a better condition than controlled blasting?

3. The BuMines has a geotomographic hazard detection system that is used to detect geologic anomalies in underground openings via electromagnetic signal propogation and computer processing of data. This system may have application for locating geologic anomalies ahead of the face which could be useful in driving entries and rooms.

***SUMMARY

This review evaluates the assumptions and interpretations of data made by the DOE. The major recommendation to expand the size of ES-II and to incorporate certain gassy mine regulations is supported with sound engineering reasoning and judgement.

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Appendix A.--Reviewers Qualifications

Project Leader: Robert J. Evans

Education: B.S. Engineering, 1964
Geneva College

B.S. Mechanical Engineering, 1968
University of Pittsburgh

M.S. Civil Engineering, 1973
University of Pittsburgh

M.S. Mining Engineering, 1985
University of Pittsburgh

Background:

Responsible for the development of improved mining equipment with primary goals being the reduction of accident rates and increased productivity in the mining industry. Identifies and defines technological impediments to advancing the Bureau of Mines' mission in mining and recommends research and development projects designed to eliminate those impediments. Specific areas of specialization include fragmentation, environmental control, coal face transportation, and shaft sinking.

Responsible for technical management of water-jet-assisted cutting program using jet pressures ranging from 1,000 to 10,000 psi; technical assistance in development of state-of-the-art for backfilling of horizontal placement holes for Nuclear Waste Disposal; and manager of program with U.S. Air Force under the Deep Basing Program which involved the technical management of five contracts and one in-house project in support of developing an egress machine. The technical management of these programs include programming, planning, operating, and controlling the activities of in-house and/or contractor personnel. Developed conceptual designs, highlighted key areas requiring additional research, outlined technical approach, and determined level of effort and funding required to accomplish the desired objective. During program execution, approved and prepared schedules, set priorities, and monitored and evaluated the progress. When required, consulted, advised, and coordinated with other personnel in the organization and various specialists on problems involved in project execution. Solved problems to include scheduling, design, manufacture, and acceptance of mechanical engineering components connected with high pressure water-jet-assisted cutting.

Reviewed suggested changes for design and construction and recommended approval or disapproval. Completed detailed specifications for procurement of complex equipment and followed up the procurement process competitive bids, quality control and final acceptance. Responsible for technology transfer of these programs by conducting symposiums, writing technical papers for publication, and oral presentations to industry.

Robert D. Schmidt, Hydrologist

U.S. Bureau of Mines, 1975-present

BS Mathematics, University of Minnesota, 1970

MS Hydrology, University of Minnesota, 1984

Selected Publications:

Schmidt, R.D., "Computer Modeling in Fluid Flow during Production and Environmental Restoration of In Situ Uranium Leaching," BuMines RI 8479, 1980, 70 pages.

Schmidt, R.D., "A Flow Model Application to Mine Dewatering," Proceedings: National Water Well Association Conference on Practical Applications of Groundwater Models, Columbus, Ohio, August 1984, pages 307-326.

Schmidt, R.D., "Facture Zone Dewatering to Control Ground Water Inflow In Underground Coal Mines," BuMines RI 8981, 1985, 84 pages.

Schmidt, R.D. and W.F. Ebaugh, "Some Considerations Regarding the Steady-State Response of Shallow Aquifers to Underground Mining," Proceedings: Symposium on Mining Hydrology, Sedimentology, and Reclamation, Lexington, Kentucky, December 1985, pages 1-7.

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Schmidt, R.D. and S.E. Follin, "Geochemical Modeling of In Situ Leaching in a Heterogeneous Porous Medium," Journal of Minerals and Metallurgical Processing, May 1987.

CHARLES D. TAYLOR
Industrial Hygienist.

Seventeen years of work experience with MSHA and the Bureau of Mines. Major experience in mining has been in evaluating and recommending improvements for the control of airborne contaminants.

- Measurement and evaluation of airborne contaminants in underground mines
- Recommendation and testing of equipment and techniques for control of airborne contaminants at the mining face.
- Researched and co-authored article on backfilling in underground high-level nuclear waste repositories.
- Participate in review of contract publications prepared for NRC
- Managed for Bureau of Mines a NRC contract to evaluate the state-of-the-art for grouting in vertical shafts.

Assistance in preparation of this Ventilation review was also provided by the following Bureau of Mines personnel:

Jon C. Volkwein - Physical Scientist, Control of methane at the working face.

Edward D. Thimons - Supervisory Physical Scientist, Cooling of underground mines.

David Hyman - Geologist, Occurrence and transport of methane in rock strata.

Jeffrey Welsh - Supervisory Physical Scientist, Mine wide monitoring systems.

Mechanical Engineer: Edwin A. Ayres

Education: B.S. Electrical Engineering
University of Pittsburgh 1969

M.S. Electrical Engineering
University of Pittsburgh 1972

B.S. Mechanical Engineering
University of Pittsburgh 1976

Background:

Responsible for developing and conducting research for underground mine hoisting systems. The major emphasis of this research has been directed towards developing inspection and evaluation techniques for determining the remaining strength of wire hoist ropes under operating conditions. Current major responsibility is for in-house wire rope testing efforts. Additional areas of investigation include: arrestment devices, wire rope terminations, hoist rope lubrication, slack rope detection, and hoisting component design. Support activities for these efforts required identification of research objectives and goals, development of procurement request documents, proposal evaluation, contract monitoring efforts, and budget and personnel requirements.